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THE FLOTATION PROCESS

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The Butte and Superior works, at Butte, Mont., where the first great flotation plant in America was installed.
(*Frontispiece.*)

THE FLOTATION PROCESS

BY

HERBERT A. MEGRAW

Mining and Metallurgical Engineer. Member of the American Institute of Mining Engineers. Formerly Member of the Editorial Staff of the "Engineering and Mining Journal." Author of "Practical Data for the Cyanide Plant" and "Details of Cyanide Practice"

SECOND EDITION

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PREFACE TO SECOND EDITION

The second edition of "The Flotation Process" was finished practically two years after the appearance of the original book. Upon the completion of the first edition, it was fully expected that a later one would be undertaken under more favorable circumstances. It was expected that litigation over patent rights would have been ended and that the theory of the process would have been established beyond conjecture. Neither consummation was reached. Patent litigation still continues, and there is no theory universally accepted.

The patent situation holds no promise of speedy settlement. As to theory, the situation is only a little better. Much work is being done and some progress made. The theory of flotation, however, is yet to be explained.

In this edition most of the material that appeared in the original book is retained. It shows the development of the process, both in theory and in practice. Some portions that were considered valueless in the light of later developments have been omitted, and considerable new matter has been inserted. The newer ideas have been brought in as completely as possible, in order to present, as truly as can be, a record of the present condition of the process.

It is hoped that the book will continue to serve as an aid to those interested in the process. As with the original edition, material has been gathered for this from all available sources and the author owes whatever value the book may have to those, named and unnamed in it, who have placed their ideas and experience at his disposal.

HERBERT A. MEGRAW.

Washington, D. C.,
July 30, 1918.

PREFACE TO FIRST EDITION

The state of the art of concentrating ores by flotation is not such that a text-book can be said to cover the field with any degree of intensiveness. The process has not been sufficiently developed to provide a basis for a work that will endure for a great length of time without careful, perhaps radical, revision. The theory upon which the process operates has not been fully elucidated, and practice certainly has not been so formulated that much forecasting can be indulged in. The probabilities are that flotation is still in its swaddling clothes and the measure of its full growth cannot be even imagined now. That its usefulness will continue to increase for a long time seems unquestionable.

To assist in the development of such an important step in metallurgy is worth any effort that may be made, and it seems that accumulation of the results of practical and theoretical investigations that have been carried out by many men at many places is an important aid to further development. My part in the furtherance of flotation is this compilation, which presents the information about the process that I have been able to acquire by practice and consultation. To simplify as much as possible has been a primary object and I have endeavored to eliminate most of the speculation that has encumbered the literature of the process up to this time, retaining only those parts that have been definitely proved correct, or can show a reasonable basis for their inclusion.

No one can claim, at this time, to know enough about flotation to pose as a master of the art. Any attempt to be complete must include the work of many individuals and the ideas of many more. I make no claim of originality for the facts put forth in this book. They have been gathered where they could be found and have been used as has seemed most appropriate. The original volume of Theodore J. Hoover has been referred to often,

and credit has been given. I owe much to that work and believe it will always remain a classic in flotation.

The reason for this book is simply that great changes have been wrought in practice and also in the theory, so that some further words seem advisable. The technical press has been freely used as a source of information, consequently much of the material has already appeared in print. There is some virtue, however, in concentrating facts so that, as with ores, one need not be burdened with material foreign to the subject.

It is impossible to thank individually the long list of operators and experimenters who have supplied material for the book. Many of them are named and references are made to their work, but much of the information has been supplied with the request that no names be mentioned as to its source. By this acknowledgment these gentlemen may know that my thanks go to them as well as to those who are specifically named.

No apologies are made for the many sources of the information that is given. To be of service, no other method is possible. If it aids operators in the advancement of a worthy metallurgical development its object will have been accomplished.

HERBERT A. MEGRAW.

New York,
August 18, 1916.

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THE FLOTATION PROCESS

CHAPTER I

CONCENTRATION BY FLOTATION

The word "concentration," when applied to the metallurgy of ore, is understood to mean any process through which the valuable constituent of the ore is segregated into a portion of less weight than the original total mass. The degree of concentration may be anything from the point where only a small amount of valueless material is taken out to the point where all the waste material is removed and nothing but the valuable material left. Exemplifying various degrees of concentration are the processes of, first, ore sorting, which is a form of manual labor, by means of which a part of the ore that may be recognized as valueless is picked out and thrown away. Ore grades may be raised much or little by this process, but as a practical system of metallurgy it is falling into disuse, because of its great cost, although largely practised in former times.

A second process of concentration is that by means of which an ore is fluxed and smelted, the concentrate containing all of the valuable metal, whether it be gold, silver, copper, lead, or whatever metal is sought, and the discarded portion containing nothing of economic worth. As a matter of fact, practically all processes of metallurgy are, in effect, concentration.

In modern metallurgy, however, we do not take such a wide interpretation of concentration. The word is usually understood to refer to those processes by means of which mechanical aid is employed to separate, as completely as possible, the siliceous gangue from a high-grade, metal-bearing mineral. Such processes almost invariably require the use of grinding machinery to prepare the ore for its mechanical separation. Grinding is carried to such point as will permit the separation of the purely mineral portions from the purely gangue portions of the ore, or if this is not wholly possible, then to approach that desideratum as closely as possible.

Concentration, as heretofore understood, referred to the separating of minerals and gangues by two classes of machinery: those that treat ores wet and those that treat them dry. In either case, the separation was accomplished by taking advantage of the difference in specific gravity of the mineral and the gangue. Minerals, carrying metals, have specific gravities that are usually higher than those of gangues, although there are exceptions to this statement. In the dry-concentrating machines, a controlled current of air is used to blow away the lighter gangue, allowing the heavier minerals to remain on a plane surface or to fall directly into appropriate receivers. In wet processes the higher specific gravity of the minerals is taken advantage of by allowing them to fall through a moving current of water, the lighter gangue material being carried away by the current and washed clear of the mineral. This is by far the most widely applied system of concentration, and a great number of machines have been devised to make use of its principles. Jigs, vanners, buddles, concentrating tables, canvas tables, blankets, riffles, cement tables, and many other devices have been used for the purpose. All of these machines are well known to the metallurgist, and it would be futile to attempt to review them here.

Coming to the new process, it seems peculiar that while for all these years we have taken advantage of gravity to aid us in separating mineral from gangue, now we have become involved with a process that, while based on the identically same principle of gravity, uses it in an exactly contrary way. In connection with it we are utilizing another well-known principle—that of surface tension. Although commercial application of the process has been accomplished only within the last few years, mill operators have, since the beginning of time, been annoyed and puzzled by the action of minerals and metals under some conditions. The amalgamator has always been bothered by his fine gold, or “greasy gold,” or tabular gold, which persisted in floating away from his amalgamated plates, despite all his efforts to gather it. The principle that is responsible for the action of gold under these conditions is the principle we are now training and making use of in flotation processes. It is really, as W. R. Ingalls has so aptly put it, “concentration upside down.” Where, in water concentration, the minerals sank and the gangue was washed away, in this process the gangue is allowed to sink and the heavy mineral is made to float away.

Operators of concentration mills have always noticed the escape of finely ground sulphides on the surface of water, and they have never been able to overcome that flotation tendency. In the concentration of molybdenite ore, the floating tendency of that mineral has sometimes made ordinary gravity concentration impossible. Even the cyanide-plant operator has often encountered a high-grade floating scum that he never could entirely abolish. The fact that it was usually rich led many operators to devise mechanical means for recovering it. Flotation, then, is merely the recognition of the existence of these tendencies, and the development and utilization of them in securing better results than could be obtained by gravity concentration.

The process is comparatively new. The chemical and physical principles upon which it depends have not been investigated with sufficient thoroughness to allow the formulation of a definite theory, but great steps have been made and we have enough to base practical work upon. It is found, however, that metallurgists must widen their reading if they desire to keep abreast of flotation. Principles of physics must be freshened up, and something must be known about colloid chemistry. Flotation concentration is not an empirical procedure from which maximum success may be obtained by mere perseverance. Some education and skill is definitely required and some excursions into the realm of pure science must be undertaken if the operator is going to learn anything or be enabled to use his judgment in a profitable way.

Flotation concentration, then, is the taking advantage of the principles of surface tension and colloid chemistry, with whatever allied principles may be involved, to separate mineral from gangue by means of floating it upon the surface of water or other solution, while the gangue is induced to sink through the surface and settle separately.

CHAPTER II

THE PATENT RECORD OF FLOTATION

The art of flotation is one of the newest efforts in metallurgical endeavor, and one that is entitled to great respect by reason of the enormous and widespread success it has attained within a comparatively short time. While no one knows just where the process, as at present understood, originated, it is possible to trace back the most important steps through the records of the patent office. Naturally, a great many important ideas never reach the patent office, but develop gradually until they appear before the public as highly perfected systems, but for one process developed in such a quiet way there are very many whose infantile efforts are recorded in the patent-office files and, it may be truly said, many of them never get any farther. A great deal of interesting information may be obtained from patent data, and a brief survey of the record, so far as it is possible to follow it, will be interesting and instructive to the student of the art.

The Patent Record of Flotation.—The first one of these that is widely known is the patent obtained by William Haynes, in 1860, upon a process for separating mineral sulphides from gangue by oil, gummy, or bituminous matter. This patent is upon a process that is not flotation, since it attempted to remove sulphides by having them adhere to a viscous oil. It is, however, worthy of record as one of the first steps in flotation, since it recognizes the tendency for sulphides to adhere or mix with oils to a much greater extent than gangue particles or oxides.

The next patent in line is that of Hezekiah Bradford, granted in 1886. Bradford's is a process based upon surface-tension factors. The flotation of sulphides on liquids is pointed out in detail, but there have been, so far as is known, no commercial applications made of the specifications as set forth in the patent. It is important, as are all such steps, since it points out a growing and widening recognition of the features that are so well known at the present time as affecting flotation. The trend of

events is perfectly clear, in that there is a development from the first agglomerating systems, which recognize only the tendency of sulphides to adhere to oils and are dependent upon viscosity to separate them from the gangue, to the bulk-oil separation processes, by means of which the buoyant property of oil is taken advantage of to float the maximum amount of mineral that the oil will carry, graduating from that into surface-tension systems, through which minerals float due to surface forces, until we finally reached the stage where it was found that gas or air bubbles would attach themselves readily to oiled mineral sulphides and tend to float, whereas the gangue exhibits a contrary tendency. The use of acids was a step that proved of great importance, since it altered the character of the liquid medium.

The Advent of Mrs. Everson.—With the development of the acid process, we come to the next important patent of the series, known as the Everson patent, granted in 1885. There is probably no figure in metallurgical history about whom there has been so much belated interest as Mrs. Everson, nor is there any one, probably, about whom so many erratic statements have been made. It will be interesting to study briefly a true account of her work in flotation, a résumé of which has been already published.*

Carrie Jane Billings Everson was born at Sharon, Mass., in 1842. She was graduated from High and Normal School and, in 1864, was married to Dr. W. K. Everson, at Chicago. Dr. Everson was an old-school physician, druggist, and chemist. Mrs. Everson first took up the study of medicine with her husband as tutor and subsequently specialized in chemistry. She never attended medical or engineering schools. She was attracted to mineralogy through Dr. Everson's investments in mining enterprises in Colorado and Mexico. Her inclination was toward whatever scientific subjects came nearest to her daily life. She was interested in medicine because her husband was a physician, and her medical knowledge was considered equal to that of Dr. Everson. She was interested in chemistry as an allied science, and became a better chemist than her husband. Mineralogy attracted her because she believed that mining would become pecuniarily profitable as well as scientifically interesting.

Mrs. Everson had a brother who was an assayer, but whose practice and research had nothing to do with mineralogy or

* *Eng. and Min. Journal*, Jan. 15, 1916.

mining, so that he gave no instructions whatever in her experiments, although it has so often been stated that Mrs. Everson's interest in flotation came about through an accidental discovery while assisting her brother in some mineralogical work. The assistance obtained from her husband was merely of a consulting nature. While he was naturally interested in mineralogy and metallurgy through his investments and his wife's research, he did not attempt the study of the subject nor bring his own mind to the practical treatment of ores.

Dr. Everson's mining investments brought him nothing but loss and practically total financial disaster. Mrs. Everson probably began her study of mineralogy as early as 1878, and about 1880 began the investigation and research that resulted in her patent in 1886, while her husband was in Mexico. The Eversons removed from Chicago to Abilene, Kan., and thence to Denver. Dr. Everson died in Denver in 1889. Mrs. Everson had given up hope of any return from their investments in mining, but she believed strongly in her invention and thought that if given proper attention the flotation process would be applied commercially and economically to the metallurgy of ores.

At the time of Dr. Everson's death, their son, John L., was about seventeen years old. Through association and tutelage the boy became practically interested in the process, and when, in 1890, Thomas F. Criley formed the partnership with Mrs. Everson, young Everson assisted him in a practical demonstration at West Cliff, in the neighborhood of Silverton, Custer County, Colo. These experiments and demonstrations were considered satisfactory in results, but so little encouragement was offered them in Colorado that it was determined to send Criley to Baker County, Ore. There was at that time, or had been just prior thereto, some mining excitement in the Sparta district, in the Powder River region. Criley put up a demonstrating plant and experimented successfully with sulphide ores of the district, and also made some demonstrations at Baker City. He died either at Baker City or Portland, and Mrs. Everson went to Oregon in the hope of straightening the tangle in which the affairs were left at the time of Mr. Criley's death. She was not a good promoter nor business woman; and nothing of practical benefit resulted from her efforts or those of Criley in Oregon.

According to her son, John L. Everson, after Criley's death, subsequent failure to interest mining companies or investors

in the oil-froth method, his mother made but little effort to continue the exploitation of this process and devoted her time to research and experiment with a dry process of flotation. In conjunction with Chas. D. Hebron, a patent was taken out about 1892. Hebron secured the financial assistance of a Mr. Pischel, also of Denver. They obtained the use of a chemical laboratory and plant in a suburb of Denver and proceeded with the experiments and demonstrations of Mrs. Everson's dry process of flotation. Hebron and Pischel got into a quarrel over money matters or division of interest, and finally Mrs. Everson gave up hope of any financial benefit from either of her processes.

Mrs. Everson went to California in 1909 to make her home with her son at San Anselmo, where she died on Nov. 3, 1914. For two or three years just prior to going to California, she taught physiology and hygiene in the State Reformatory for Girls at Morrison, Colo. From this might have been obtained the impression that the inventor of flotation process was a school teacher. Mrs. Everson had never taught school at the time her flotation experiments were made, although she was graduated from a normal school and was prepared to take up teaching as a profession. Her marriage turned her mind and her talent toward medical and chemical science, and thence, naturally, in view of their mining investment, to mineralogy and metallurgy. That she did not receive any benefit from her discovery was due primarily and largely to the loss of her husband's accumulated earnings, but in no small degree to the fact that as a metallurgist she was a quarter of a century in advance of her profession.

The Everson Patent.—The original Everson patent granted in 1885 specified a process for separating sulphide in an ore from gangue. The selective action of oil for sulphides, discovered by Haynes, was utilized in her invention, but she discovered in addition that acid added in small quantity to the pulp greatly increased the selective action of the oil, and the oiled mineral could be separated from the gangue by giving the mass a thorough agitation and allowing the sulphide and oil to float to the surface. The amount of oil used is variously stated in her experiments, varying from 6 to 20%, and the amount of acid was usually less than 1%. In considering the Everson patent, it must be carefully borne in mind that none of the amounts of oil specified by Mrs. Everson were enough to float sulphide purely by reason of the buoyant properties of the oil, but were only

sufficient to provide a froth that would enable the sulphides to be carried to the surface and floated. It will be of interest to review here a copy of the original Everson patent.

CARRIE J. EVERSON, OF CHICAGO, ILLINOIS

Process of Concentrating Ores

Specifications forming part of Letters Patent No. 348157, dated August 24, 1886.

Application filed August 29 1885. Serial No. 175665. (Specimens.) To all whom it may concern:

Be it known that I, Carrie J. Everson, of Chicago, in the county of Cook and State of Illinois, have invented a certain new and useful improvement in Processes of Concentrating Ores; and I do hereby declare that the following is a full, clear, and exact description thereof.

The discovery which forms the basis of my invention is that metals and metallic substances in a comminuted state will unite with compounds of fats or oils and acids, and that such compounds will not unite with comminuted quartz or other rocky gangue. The essential feature of the method which constitutes my invention, therefore, consists in commingling with pulverized ore a fat or an oil, either animal, mineral, or vegetable, or a fatty constituent or acid of an animal or vegetable fat or oil, or any constituent of a mineral oil, together with an acid, either mineral or vegetable, or a soluble neutral or acid salt, for the purpose of effecting a union of the free metal or metallic portion of the ore with such admixed material, whereby the same may be retained in the subsequent separation of the quartz or other rock therefrom by washing or other suitable means.

The invention is more specially applicable to the treatment of ores in which the metal or metallic portion is mixed with quartz or other rock as distinguished from the mixture of mineral with clay, though it is applicable to ores containing alumina, together with quartz or other rock, to the extent of permitting the removal of the silica or sand.

The operation of concentration as a whole, or in its practical form as applied to the separation of rocky mineral ores, involves the reduction of the quartz or other rock containing the mineral to a powder, the addition thereto of the fat or oil and acid or salt, and the subsequent removal of the gangue.

If my invention is inapplicable to any particular rocky ores or class or classes of rocky ores, or to the concentration of any particular metal, such limitations are not now known to me. Its commercial value, however, will probably be restricted to its use in connection with ores bearing the precious metals, such as gold, silver, and copper.

Among the ores upon which I have operated successfully by means of my invention are the following: Ores containing native gold and silver, kerargyrite, argentite, argentiferous galena, and a variety of double and otherwise compound sulphides of silver, with copper, antimony, arsenic, and other base-metal sulphides. I have also operated successfully upon pyritic ores, white, yellow, green, and intermediate shades, having variable constituents of iron, copper, arsenic, and sulphur, and mostly auriferous, and having variable proportions of such constituents. I have also operated successfully upon ores containing tellurides of gold, silver, and lead, and others containing the oxides and carbonates of copper and the carbonate of lead. All the metals or metallic mineral portions of these ores are acted upon by the admixture described, in such manner as to permit the rocky gangue to be removed by a washing process, after which several metals may be separated by the usual means.

In putting my invention into practice any fat or oil, and any acid, either mineral or vegetable, or any soluble neutral or acid salt, or any compound of fats and oils with appropriate acids, may probably be successfully employed, at least such is the case with all of these agents with which I have so far experimented. I have used petroleum and one of its several constituents—namely, paraffine-oils—also tallow (melted), lard, lard-oil, red-oil (impure oleic acid), cotton-seed oil, castor-oil, sperm-oil, and linseed-oil, and some combinations of these with each other. The acids which I have employed are sulphuric, hydrochloric, nitric, phosphoric, acetic, oxalic, tannic, and gallic. I have also used the following salts, to wit: the sulphates and chlorides of sodium, zinc, and copper, and the double sulphate of potash and alumina. The selection of the appropriate agents will, however, be largely determined in the practical working of my invention by the consideration of economy, which will obviously exclude the greater number of those above enumerated. A reasonably cheap and practicable use of my invention will be as follows, the quantity named being suitable for laboratory use and the character of the ore specified being substantially that of an ore upon which I have successfully operated by the preparation and in the manner hereinafter described.

Take an ore assaying twelve ounces per ton in silver and containing forty-eight per cent. silica, 6.3 per cent. zinc, 1.5 per cent. copper, fifteen per cent. iron and aluminium, 6.5 per cent. lead, 14.18 per cent. sulphur, 7.19 per cent. arsenic. Of this ore take four (4) ounces by weight in pulverulent form, prepare a mixture containing sulphuric acid, cotton-seed oil, and water, in all about twelve fluid drams of which ten drams are of water and about two drams are acid and oil in the proportions of fifteen parts of the oil and two parts of the acid by measure. In making this fluid mixture the acid and oil are first mixed with each other, the acid being added to the oil very gradually, so that the tem-

perature will not rise above 120° Fahrenheit. The stirring in of the acid should be thorough, as it tends at first to gravitate to the bottom. After a few hours, in a summer temperature, the mixture will be ready for use, and, preferably, in such a temperature should not be prepared long before using, though if it should have stood long enough to solidify it should be gently heated before adding the water thereto. In winter or in air-tight vessels it may be kept for a number of weeks or even longer, and then rendered fit for mixing with water by heating gently when required for use. The water may be advantageously added in installments of about three equal parts, and the mixture stirred after each addition of water until it stiffens. After stirring in the entire quantity of water the compound is added to the ore, the proportions of ore and compound being chosen with a view to producing a stiff mass after the materials have been incorporated, such proportions being therefore variable in different cases with this end in view. The stirring or incorporation of the ore with the liquid should of course be thorough for the purpose of bringing the mineral into contact with the oil and acid as completely as possible, and after such incorporation the mass is then in condition for the washing out of the quartz by the action of water which will be applied to the mass in sufficient quantities for this purpose. The washing should promptly follow the mixing, and in this operation the mass be opened out or broken up and thoroughly stirred in the water, in order that the sand or quartz may be freed and carried away. In treating so small a bulk as above specified the mass may be squeezed repeatedly in the hand in a basin of water, the substance so manipulated being expressed between the fingers each time it is squeezed, and thus made to expose new surfaces to the water from which the sand will be detached, so as to fall to the bottom of the basin. In practice upon large masses any vessel having an outlet or outlets as its bottom for the escape of the water and sand will be suitable for this operation of washing, and mechanical means will of course be employed to break up the mass. The concentrated mineral will accumulate in a pasty mass or lump or lumps and will contain the metallic portion of the ore, together with the hydrated oil and acid, which latter may be removed by heating and afterward roasting, or by other suitable means. The use of petroleum or of a constituent thereof, either by itself or in combination with tallow, (heated), cotton-seed oil, or other fat or oil, will be even less expensive.

When petroleum or a constituent thereof is used the oil should desirably be first mixed with the ore, then water added containing a suitable amount of free acid or a soluble neutral or acid salt, the quantity of water being ample for the washing-out operation, which is to follow, and the quantity of acid sufficient to cut the sand away from the otherwise cohering mass. In the case of petroleum or its constituent, paraffine-oil, one or two fluid drams of acid to one gallon of water is

sufficient for this purpose. The petroleum which I have used was 30° Baumé, and I have found three fluid drams of oil abundant for properly moistening two ounces of heavy ore, or in the ratio of about a barrel of oil to the ton of ore, the amount being, of course, variable with the relative bulkiness of the ore.

In the use of petroleum, or of a liquid constituent thereof, like paraffine-oil, the condition of the concentrated mass is more liquid than when a vegetable or an animal oil or a fatty constituent thereof is used, and a somewhat different means or method should be employed for removing the sand. In practice, the concentrate, after thorough agitation of the mass and detachment of the sand, will in this case be preferably removed by means of a constant overflow of water from a washing-out vessel, by which overflow the concentrate will be floated off. Devices and methods now well known in wet separation of ores will be suited to this part of the operation, bearing in mind that the sand and mineral are merely transposed or their relative positions are reversed, because the sand is heavier than the mixture of mineral, oil, and acid. A proper selection of devices for this purpose will be apparent to those skilled in the wet separation of ores. After removing the quartz or rock by washing or any suitable means the mineral may be roasted, in which operation the water, if present, will be dispelled, the oil will burn out, and the acid will be decomposed and eliminated, in the case of sulphuric-acid, which will commonly be used, by conversion into sulphurous-acid gas, which passes off. In case fixed oils are used the mass may be allowed to stand until the water has run off. To hasten the operation of removing the water the mass may be heated, say, to about 212° Fahrenheit or less, whereupon the water will generally be promptly freed and may be poured off or it may be evaporated. The operation of smelting is facilitated by the presence of the fat acting simply as a fuel.

The disposition made of the mineral or metal after concentration, whether by roasting and subsequent smelting or by smelting without previous roasting, or otherwise, belongs to the after treatment and is not material to my present invention. The proportions in which the acids or salts are added to the oils or fats may vary according to the kind of acid or oil employed, and also according to the kind of ore to be treated; and the manipulation of the substances employed may be varied from that above set forth in either formula given. These matters may be determined in individual cases by the operator; and I do not, therefore, restrict myself to any particular proportions of the substances employed, though I have above indicated proportions of certain acids and oils by one or the other of which practical results may be obtained upon almost all varieties of ores.

It is also not essential to my invention that the acid or salt employed with a vegetable oil be added to the oil before the incorporation of the oil with the ore, as it is entirely practicable, at least in most, and pos-

sibly in all, cases, to first mix such oil with the ore and thereafter add the acid, as set forth in the use of petroleum.

X I am aware that it has been proposed in a patent to Tunbridge, No. 228,004, dated May 25, 1880, to recover finely comminuted metal held in suspension in water by the use of soap or a saponaceous compound formed by a partial or imperfect blending of an alkali or an alkaline salt with a fat or an oil. The action of the soap or saponaceous compound when diffused through hard water containing the metal in suspension is to form with the lime salt in the water a coagulum in which the particles of metal are enveloped and by which they are held during a subsequent removal of the water. When the water is soft a lime salt must be added to form the coagulum. I disclaim the use of any substance in connection with the fats and oils capable of saponifying them. I also disclaim the method set forth in said Tunbridge patent as being entirely dissimilar to that herein claimed. Said Tunbridge method is stated in said patent to consist "in first depriving the water of as much earthy non-metallic matter as possible," this being effected by any of the ordinary methods in use for that purpose, such as letting it stand until the earthy matter precipitates. My invention, on the other hand, has reference solely to the separation of metal or metallic mineral from earthy non-metallic matter (rocky gangue), and not to its subsequent separation from water. In the practice of my method, moreover, the metal or mineral and non-metallic matter (rocky gangue) are together and in the first instance mixed with the fat or oil, and the non-metallic matter is afterward separated from the metal or mineral by the peculiar action of the acid or soluble neutral or acid salt operating in the presence of water to detach the gangue from the mass.

With the understanding that I am not restricted in the matters mentioned, I claim, broadly—

1. In the separation of pulverulent ores containing rocky gangue, the method of treatment herein described, which consists in mixing with such pulverulent ore a fat or an oil or a constituent thereof, an acid or soluble neutral or acid salt, and water, finally breaking up the mass to allow the sand to separate therefrom.

2. The method, substantially as described, of separating metals or metallic minerals from rocky gangues, which consists in mixing a fat or an oil or a constituent thereof with pulverized ore and washing out the gangue with water containing an acid or a soluble neutral or acid salt.

In testimony that I claim the foregoing as my invention I affix my signature in presence of two witnesses.

CARRIE J. EVERSON.

Witnesses:

M. E. Dayton,

G. F. Lanaghen.

The original Everson process was improved in 1892 by the patent of Hebron & Everson, in which pulverized ore was mixed with a buoy-stock in a dry state and then the prepared ore was applied to a stratifying apparatus into which air could be introduced for the purpose of producing a froth.

Robson & Crowder Process.—During the life of these processes, and for some time afterward, little was heard of flotation and practically no economical advantage was taken of the principles involved. In 1894 Robson & Crowder brought out a process for separating the gangue and sulphides, but they made use of oil selection only. Their invention was very little, if any, different from the ones that had gone before. In 1898, F. E. Elmore invented the bulk-oil process which has always borne his name. In this process the total ore pulp was mixed, in water solution, with a large bulk of oil. The oil was used in sufficient quantity to provide a floating or buoyant effect, by means of which the minerals were to be separated from the gangue. The idea was that the mineral sulphides, having a tendency to become easily oiled, would be caught and held in the oil sheet, while the gangue would not become oiled and would sink through the solution to the bottom of the containing vessel. This process was designed to be carried out in a slowly revolving tube or drum, by means of which the ore particles could be brought into contact with the oil cover without seriously disturbing or emulsifying the oil. Naturally, the oil used must have been in large quantity—several times the weight of the ore itself. After the mixing process, separation was carried out in a spitzkasten or similar device. The process has had quite a large application in various parts of the world, but at the present time it has been discarded in favor of more economical processes. The bulk-oil process used a large quantity of oil, and there was some loss which could not be easily recovered. The process forms an important step, since it awakened interest in oil flotation and led to further important developments.

In 1902 Van Meter & Boss invented an oil process somewhat similar to the Everson, and although published results were good, nothing was ever done commercially with it beyond two installations in Peru.

Discoveries of Potter and Delprat.—An important invention in 1902 was that of Chas. P. Potter, who accomplished flotation in a 1 to 10% acid solution. By combining equal weights of ore

and acid solution and heating the mixture, sulphides were caused to rise to the surface from which they were continuously skimmed off. The function of the gas was not mentioned but its actual necessity was undoubtedly realized. About the same time the Delprat process was invented by Guillaume D. Delprat, of Broken Hill, Australia, who accomplished the same results obtained by Potter, but by using salt-cake instead of acid. In each case bubbles of gas were generated which served to float the sulphides. Potter received priority of patent, though the process is usually known as the Potter-Delprat system. Both of these inventors proceeded to develop their processes and, from time to time, patented improvements. They soon found that a small quantity of oil was an advantage and made use of it.

Froment's Oil-Froth Discovery.—A discovery, which may be called of the greatest possible importance in flotation, was made by Alcide Froment, of Traversella, Italy, in 1901. He brought out the fact that, when small quantities of oil and acid are agitated in water with ore, the sulphide particles become preferentially oiled and attach themselves to gas bubbles and float to the surface, the gangue particles declining to receive the oil and sinking to the bottom of the solution. Froment desired to increase the quantity of gas, and to do this added small quantities of calcite, which, in combination with the acid, developed carbonic-acid gas, with which the bubbles were filled. The importance of this discovery cannot be overestimated, since most of the modern and largely used processes are based on the preference of oiled sulphides to adhere to bubbles and rise to the surface, while the gangue acts in a contrary manner.

In about 1903 the Cattermole process was invented. This was not a flotation process, but made use of oil, and for this reason is interesting enough to be considered in connection with oil-flotation processes. In the Cattermole process a quantity of oil varying from 4 to 6% was added to a flowing pulp, producing a tendency to oil the sulphides and make them stick together, forming large and heavy globules or granules. These granules were heavy enough to fall to the bottom and remain in a pulp current while the gangue was washed away. In this process we have the spectacle of an oil-concentration system which was not flotation, but the exact reverse—in other words, gravity concentration carried out by a new method.

Another pair of inventors, Goyder & Laughton, brought out

the underlying basis of the Potter-Delprat acid flotation system in 1903. Their invention was a machine instead of a process, but it was not very largely used, although it was tried at several plants in the Broken Hill district of Australia. In 1903 Jacob D. Wolf brought out another method of applying flotation principles. He used an oil to secure a high concentration of low-grade concentrates in the first step and by washing this concentrate with hot water to increase subsequently its grade. He also used steam to blow into the tailings, which raised to the surface globules of oil that had been lost, and with them some sulphide was also recovered. No practical application of the patent has been made so far as is known.

The Elmore Vacuum Process.—In 1904, Francis E. Elmore figured again in the oil-flotation field through the patent he secured for a system of vacuum oil-flotation. In this, flotation was accomplished by raising the mineral to the surface of the pulp in a froth and the froth was formed by the addition of a small quantity of oil and by liberation of the air in the pulp itself in a fine state of subdivision. The liberation of this air was accomplished by applying a vacuum to the pulp and warming it, by which means the air dissolved was released and, attaching itself to the slightly oiled mineral particles, raised them to the surface. It may be noted that about 2.2%, by volume, of air is dissolved in water at normal temperature and pressure. When the pressure is lowered and the temperature raised, that air is released, forms bubbles which serve to form a froth just as in the agitation processes. The amount of air self-contained in the water may be computed to have a certain definite lifting power, and in the development of the probable possibilities the theoretical indication is that 360 lb. of the Broken Hill, Australia, zinc-lead sulphide may be raised to the surface by the amount of air contained in a pulp consisting of 1 ton of ore to 6 tons of water. These figures are given by T. J. Hoover in "Concentrating Ores by Flotation," page 102. It may be noted, however, that in actual practice the weight of sulphide that is floated is considerably greater than the theoretical capacity based on the efficiency of air released from solution in the water. Undoubtedly some of this additional work is done by the gases liberated by the action between acid and carbonate, but there is an additional reason for the discrepancy, and this matter will be discussed in the chapter dealing with the

theory of the process. The quantity of oil used in the Elmore process was gradually reduced to about 10 lb. per ton and finally to 3 lb. per ton of ore. The machine, to which the success of the process is largely due, was an ingenious device designed and built by Elmore himself. It was used with notable success at the Sulitelma copper mine, in Norway.

The De Bavay process is one in which a thin film of flowing pulp is brought to the surface of a vessel of water where advantage is taken of the surface tension of the liquid and the sulphide is floated. This process was patented by Auguste J. F. De Bavay in 1904. De Bavay mentions the fact that if the surface of the sulphide particles is coated with a film of carbonate from weathering, this is a detriment to the process and should be removed by soaking the ore in a weak solution of ammonium carbonate. The same effect can be had by passing carbonic-acid gas through the pulverized wet ore, or by so grinding the ore that the undesirable surface shall be removed by attrition. This process was installed in one or two places in Australia and is said to have been a success. The method has, since its inception, undergone a great many changes, and its users are now employing small quantities of oil to assist in extraction.

Another patent, in 1906, that of James H. Gillies, was applied on a commercial scale in Australia. Although this process makes use of the principles of Potter and Delprat, the apparatus was unique. The plant was a failure economically and the process has been abandoned.

The Minerals Separation Process, as it may be called here, is embodying in the famous patent that bears U. S. No. 835,120 and dated Nov. 6, 1906. This was issued to Sulman, Picard and Ballot and covered the use of small quantities of oil "less than 1% upon the dry weight of the ore treated," in connection with peculiar degrees of agitation and claimed to be characterized by the character of the froth produced.

This is the patent upon which the Minerals Separation Corporation bases its claims to preëminence in the field of flotation in the United States, and upon the equivalent British patent its claims in Great Britain. As to the validity of patent and the basis of the claims founded upon it, no discussion can take place here. To the technician the various details of the patent seem to be developments of points that had been known or pointed

to before. Because of the great interest in the definite terms of the patent, however, it is reproduced here.

To all whom it may concern:

Be it known that we, HENRY LIVINGSTONE SULMAN, HUGH FITZALIS KIRKPATRICK-PICARD, and JOHN BALLOT, subjects of the King of England, residing in London, England, have invented certain new and useful Improvements in Ore Concentration, of which the following is a specification.

This invention relates to improvements in the concentration of ores, the object being to separate metalliferous matter, graphite, and the like from gangue by means of oils, fatty acids, or other substances which have a preferential affinity for metalliferous matter over gangue.

In the process described in the previous United States patent, No. 777,273, granted to A. E. Cattermole, an amount of oil varying from four per cent. to six per cent. of the weight of metalliferous matter present is agitated with an ore pulp, so as to form granules which can be separated from the gangue. In the previous United States patent, No. 777,274, granted to A. E. Cattermole and others, a similar method of separation is employed, oleic acid being produced *in situ* in the ore pulp.

We have found that if the proportion of oily substance be considerably reduced—say to a fraction of one per cent. on the ore—granulation ceases to take place, and after vigorous agitation there is a tendency for a part of the oil coated metalliferous matter to rise to the surface of the pulp in the form of a froth or scum. This tendency is dependent on a number of factors. Thus the water in which the oiling is effected is preferably slightly acidified by adding, say, a fraction of one per cent. up to one per cent. of sulfuric acid or other mineral acid or acid salt, the effect of this acidity being to prevent gangue from being coated with oily substance, or, in other words, to render the selective action of the oil more marked; but it is to be understood that the object of using acid in the pulp according to this invention is not to bring about the generation of gas for the purpose of flotation thereby, and the proportion of acid used is insufficient to cause chemical action on the metalliferous minerals present. Again, we have discovered that the tendency for the oily substance to disseminate through the pulp and the rapidity with which the metalliferous matter becomes coated is increased if the pulp is warmed. The formation of froth is assisted by the fine pulverization of the ore, and we find that slime mineral most readily generates scum and rises to the surface, while larger particles have less tendency to be included in the froth. The proportion of mineral which floats in the form of froth varies considerably with different ores and with different oily substances, and before utilizing the facts above mentioned in the concentration of any particular ore a simple preliminary test is necessary to determine which oily substance yields the proportion of froth or scum desired.

The following is an example of the application of this invention to the concentration of a particular ore. An ore containing ferruginous blende, galena, and gangue consisting of quartz, rhodonite, and garnet is finely powdered and mixed with water containing a fraction of one per cent. or up to one per cent. of a mineral acid or acid salt, conveniently sulfuric acid or mine or other waters containing ferric sulfate. To this is added a very

small proportion of oleic acid (say from 0.02 per cent. to 0.5 per cent. on the weight of ore). The mixture is warmed, say, to 30° to 40° centigrade and is briskly agitated in a cone mixer or the like, as in the processes previously cited, for about two and one-half to ten minutes, until the oleic acid has been brought into efficient contact with all the mineral particles in the pulp.

When agitation is stopped, a large proportion of the mineral present rises to the surface in the form of a froth or scum which has derived its power of flotation mainly from the inclusion of air-bubbles introduced into the mass by the agitation, such bubbles or air-films adhering only to the mineral particles which are coated with oleic acid. The minimum amount of oleic acid which can be used to effect the flotation of the mineral in the form of froth may be under 0.1 per cent. of the ore; but this proportion has been found suitable and economical.

If the ore were crushed to ninety mesh to the linear inch (half of which ore will pass through one hundred and fifty mesh sieve), the froth may contain about seventy per cent. to eighty per cent. of the metalliferous matter present in the ore. This froth is removed from the pulp by spitzkast, upcast, skimming, draining, or otherwise. After subsidence the oil-coated metalliferous matter removed as froth is separated from any liquid which may have accompanied it and treated with a dilute solution of caustic alkali, which removes the oleic acid in the form of a solution of soap.

If desired, the oleic acid used in the first instance may be produced *in situ* in the pulp by decomposing a dilute soap solution with mineral acid, as described in the previous patent, No. 777,274, cited above. The oleic acid or other fatty acid forming the coating on the metalliferous matter which produces the froth may give rise to insoluble soaps on the surface of the metalliferous matter if soluble lime, iron, or other salts are present in small quantity during the production or on the breaking down of the froth with alkali. Such insoluble soaps are difficult to remove and are capable of adhering to air and causing flotation, much the same as the fatty acids do.

The metalliferous matter which did not form part of the froth (generally the larger particles) remains in admixture with the gangue in the pulp. To recover this, the pulp is distributed in a thin layer on a shaking-table, convex buddle, or the like, whereon the mineral is exposed to a free-air surface, which exposure may be increased by the application of air-blast or air-jets or the like and thereafter brought onto the edge or surface of liquid, whereby the metalliferous matter floats and is separated from the gangue, which sinks, as described in the specification of our previous United States application Serial No. 246,637, filed February 20, 1905.

The proportion of mineral recovered in the froth and that recovered by table flotation may be considerably varied; but, generally speaking, the froth will separate the slime mineral while the larger particles are recovered by the latter method.

In the accompanying drawings, Figure 1 is a diagram in perspective illustrating one form of apparatus suitable for carrying this invention into practice, and Fig. 2 represents in perspective an apparatus for carrying out a secondary step in the process.

Referring to Fig. 1, a mixing vessel A (of which there may be any number

in series) is provided with a rotatable stirrer B. Crushed ore is fed from a hopper C into the vessel by a band D. A pipe E, controlled by a tap E', delivers circuit-water to the vessel, and oleic acid or other oil is introduced through the pipe F and tap F'. The outer cock G from the vessel A communicates through a swanneck pipe H with the froth separating apparatus. In passing from the frothing apparatus A to the spitzkasten (say between O and K) the pulp may, if desired, be run in a thin layer over a smooth slightly inclined plane.

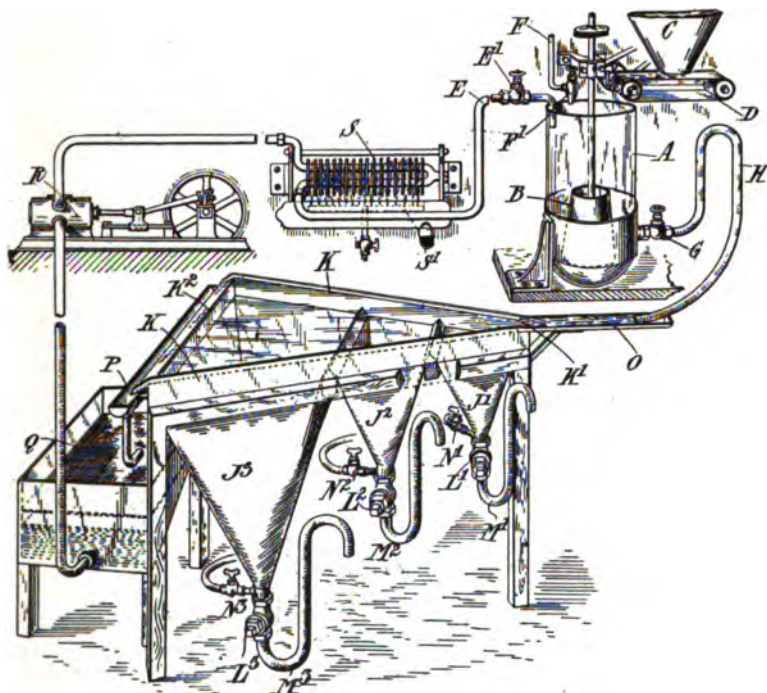


FIG. 1.—Sulman, Picard and Ballot flotation process.

The froth-separating apparatus comprises several (say three) pointed boxes J' J² J³, which open at the top into a horizontal channel consisting of side walls K. The channel has a narrow inlet K' and spreads out to a wide outlet K². The pointed boxes J' J² J³ have fullway-cocks L' L² L³ at the bottom, leading to swan-neck discharge-pipes M' M² M³. An upcurrent of water may be led in at the bottom of each box through a tap N' N² N³.

The boxes are all filled with circuit-water. The pulp from the vessel A is distributed horizontally from the flat trough O through the inlet K'. The heavy sands and coarser particles of mineral sink into the first box J', from which they are led to a shaking-table, convex buddle, or the like, to be treated as above described. The middlings or medium sands

fall into the box J^1 , and if they contain any mineral may be removed for further treatment by agitation. The upcurrent of water from the taps $N' N^2$ prevents the deposition of any slime in these boxes. The fine sands or gangue slimes settle in the last box J^3 , from which they are discharged to waste or further treatment.

The slime mineral in the form of froth or scum floats from the liquid and is carried by the stream over the outlet K^2 into a launder P and thence to a filter Q , where the metalliferous matter is removed from the circuit-water, which is returned to the vessel A by a pump R . The circuit-

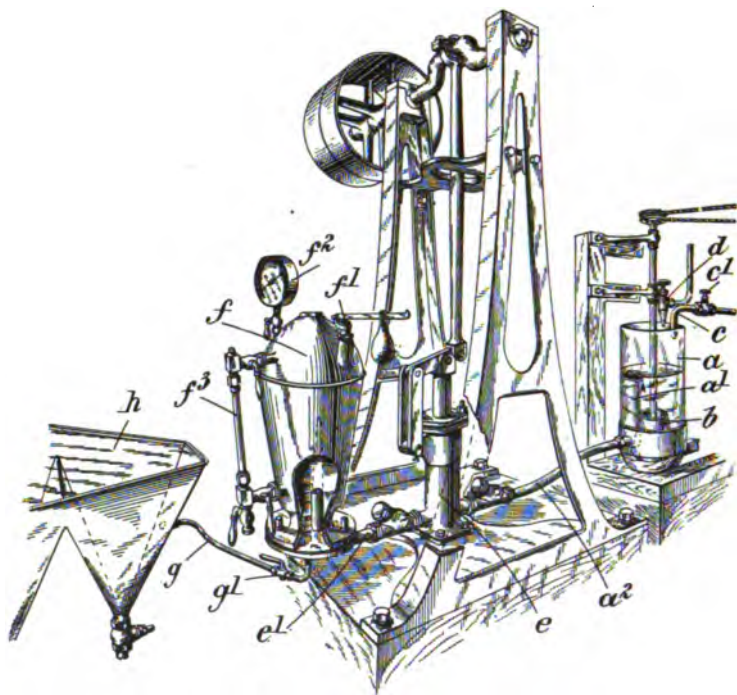


FIG. 2.—Alternative apparatus of Sulman, Picard and Ballot.

water may be brought to the proper temperature by passing it through a heater S , having a burner S' , before admitting the water to the vessel A .

An alternative method for the recovery of any sunk oiled metalliferous matter which may be deposited in the second and third spitzkasten is as follows: The products suspended in circuit liquor are removed from the spitzkasten and placed in a vessel in which they are submitted to an additional pressure of air or other gas of from, say, one to two atmospheres or over. On relief of such pressure the bubbles of air or other gas so generated throughout the mass at once sweep to the surface thereof all the metalliferous matter in the form of a froth which can be separated as before. This idea

is not claimed broadly in this case, but forms the subject-matter of an application filed by us on January 9, 1906, Serial No. 295,326.

Referring to Fig. 2, a mixing vessel *a* (of which there may be several in series) is provided with a rotatable stirrer *b*. Crushed ore or similar finely divided mineral is fed into the vessel *a*. A pipe *c*, controlled by a tap *c'*, delivers circuit-water to the vessel, and in cases where oil is used the oil is introduced through the pipe *d* in quantity sufficient to produce a thin coating of oil on these mineral particles for which oil has an affinity.

The pulp mixed with oil escapes over the lip of the discharge-conduit *a'* and passes through the pipe *a''* to a pump *e*. Hence the pulp is pumped through discharge-pipe *e'* into the closed chamber *f*, which is constructed to withstand a considerable internal pressure and is provided with a safety-valve *f'*, the pressure-gage *f''*, and a gage-glass *f'''* to indicate the level of the pulp in the chamber. An outlet-pipe *g*, having a cock *g'*, leads to a series of spitzkasten *h*, filled with circuit-water.

The operation is as follows: The cock *g'* is closed. Pulp is pumped into the chamber *f*, which contains air or other gas, and the pumping is continued until the pressure in the chamber rises to, say, fifty to one hundred pounds per square inch. The pressure is sufficient to cause the air or other gas to be dissolved to a considerable extent in the pulp. After the lapse of a few minutes for the due solution of the compressed air or a portion of it by the pulp or the liquid the cock *g'* is opened and the pulp is discharged into the open spitzkasten *h*, where the liquid is of course under atmospheric pressure. The pulp *e* may be stopped during this discharge. The whole of the mineral to which air bubbles are attached—say the oiled mineral—at once rises to the surface as a coherent scum or froth. A surface current of water is maintained in the spitzkasten, and the floating material is thus removed and separated from the gangue, which remains sunk or suspended in the liquid.

The nature and arrangement of the apparatus used may be varied without departing from this invention.

What we claim as our invention, and desire to secure by Letters Patent, is—

1. The herein-described process of concentrating ores which consists in mixing the powdered ore with water, adding a small proportion of an oily liquid having a preferential affinity for metalliferous matter (amounting to a fraction of one per cent. on the ore), agitating the mixture until the oil-coated mineral matter forms into a froth, and separating the froth from the remainder by flotation.

2. The herein-described process of concentrating ores which consists in mixing the powdered ore with slightly acidified water, adding a small proportion of an oily liquid having a preferential affinity for metalliferous matter (amounting to a fraction of one per cent on the ore), agitating the mixture until the oil-coated mineral matter forms into a froth and separating the froth from the remainder by flotation.

3. The herein-described process of concentrating ores which consists in mixing the powdered ore with slightly acidified water, adding a small proportion of an oily liquid having a preferential affinity for metalliferous matter (amounting to a fraction of one per cent on the ore), warming the mixture,

agitating the mixture until the oil-coated mineral matter forms into a froth and separating the froth from the remainder by flotation.

4. The herein-described process of concentrating ores which consists in finely powdering the ore, mixing it with slightly-acidified water, adding a small proportion of an oily substance having a preferential affinity for metalliferous matter (amounting to a fraction of one per cent. on the ore), warming the mixture, agitating the mixture until the oil-coated mineral matter and into a froth, separating the froth from the remainder by flotation, and removing the oily coating from the mineral.

5. The herein-described process of concentrating ores which consists in mixing the powdered ore with water, adding a small proportion of oleic acid amounting to 0.02–0.5 per cent. on the ore, agitating the mixture until the oleic acid has been brought into efficient contact with the mineral and has formed a froth therewith, and separating the froth from the remainder by flotation.

6. The herein-described process of concentrating ores which consists in mixing the powdered ore with water containing a fraction of 1% of sulfuric acid, adding a small proportion of oleic acid amounting to 0.02–0.5 per cent. on the ore, agitating the mixture until the oleic acid has been brought into efficient contact with the mineral and has formed a froth therewith, and separating the froth from the remainder by flotation.

7. The herein-described process of concentrating ores which consists in mixing the powdered ore with water containing a fraction of 1 per cent. of sulfuric acid, adding a small proportion of oleic acid amounting to 0.02–0.5 per cent. on the ore, warming the mixture to 30°–40° centigrade, agitating the mixture until the oleic acid has been brought into efficient contact with the mineral and has formed a froth therewith, and separating the froth from the remainder by flotation.

8. The herein-described process of concentrating ores which consists in finely powdering the ore, mixing it with water containing a fraction of 1 per cent. of sulfuric acid, adding sufficient oleic-soap solution to produce oleic acid amounting to 0.02–0.5 per cent. on the ore, warming the mixture to 30°–40° centigrade, agitating the mixture until the oleic acid has been brought into efficient contact with the mineral and has formed a froth therewith, separating the froth from the remainder by flotation, filtering off the froth and removing the oleic acid therefrom by treatment with an alkali.

9. The process of concentrating powdered ores which consists in separating the mineral from the gangue by coating the mineral with oil in water containing a small quantity of oil, agitating the mixture to form a froth, and separating the froth.

10. The process of concentrating powdered ores which consists in separating the mineral from the gangue by coating the mineral with oil in water containing a small quantity of oil, warming the mixture, agitating the mixture to form a froth, and separating the froth.

11. The process of concentrating powdered ores, which consists in separating the mineral from the gangue by coating the mineral with oil in water containing a small quantity of oil, and a quantity of acid insufficient to cause chemical action on the metalliferous minerals present, agitating the mixture to form a froth, and separating the froth.

12. The process of concentrating powdered ore which consists in separating the minerals from gangue by coating the minerals with oil in water containing a fraction of 1 per cent. of oil on the ore, agitating the mixture to cause the oil-coated mineral to form a froth, and separating the froth from the remainder of the mixture.

13. The herein-described process of concentrating ores which consists in finely powdering the ore, mixing it with water containing less than one per cent. of sulfuric acid, adding sufficient oleic-soap solution to produce oleic acid amounting to 0.02–0.5 per cent. on the ore, warming the mixture to 30°–40° centigrade, agitating the mixture until the oleic acid has been brought into efficient contact with the mineral and has formed a froth with the finer mineral, distributing the mixture on the surface of a current of water running over columns of water, so that the coarser minerals and sands, the finer sands and the gangue slimes successively deposit out while the froth is floated away by the current, filtering off the froth and removing the oleic acid therefrom by treatment with an alkali and separating the coarser mineral from the sands by exposing them alternately to air and water.

In testimony whereof we have signed our names to this specification in the presence of two subscribing witnesses.

HENRY LIVINGSTONE SULMAN.

HUGH FITZALIS KIRKPATRICK-PICARD.

JOHN BALLOT.

Witnesses:

GEO. J. B. FRANKLIN,

T. J. OSMAN.

Disclaimer in Letters Patent No. 835120.

DISCLAIMER

835120.—*Henry Livingstone Sulman, Hugh Fitzalis Kirkpatrick-Picard, and John Ballot*, London, England. ORE CONCENTRATION. Patent dated November 6, 1906. Disclaimer filed March 28, 1917, by the assignee, *Minerals Separation, Limited*.

"Your petitioner, therefore, for the purpose of complying with the requirements of the law in such case made and provided, and of disclaiming those parts of the thing patented which your petitioner does not choose to claim or hold by virtue of said Letters Patent No. 835120, does hereby disclaim from claims 9, 10 and 11 of said Letters Patent No. 835120, any process of concentrating powdered ores excepting where the results obtained are the results obtained by the use of oil in a quantity amounting to a fraction of one per cent. on the ore."

[*Official Gazette*, April 3, 1917.]

The Macquisten Tube Process.—In the chronology of the important patents, we now come to that of Arthur P. S. Mac-

quisten, taken out in 1904, in which he set forth a process and tube apparatus which made use of surface tension for separating the minerals, whereby some of them floated and some sank. The apparatus was new and important, but the principle upon which it depended was the well-known one of surface tension. The process was first installed at the Adelaide Reduction Works, Golconda, Nev. In 1911, the Federal Mining & Smelting Co. installed 119 of the Macquisten tubes at the Morning Mill, at Mullan, Ida., for treating middlings containing blende, galena, siderite and quartz, and the results, while not altogether satisfactory, were of sufficient value to insure the continuous use of the tubes, and they are running to this day and doing good work. The capacity of the installation is about 150 tons per day.

Murex Patents.—The next important process is that known as the Murex, invented in 1908 by Lockwood and Samuel, and is a process for making sulphide particles more magnetic by utilizing the selective action of oils. In 1909 they described their method of increasing the selective action of oil by treating it with various soluble salts, and also a process through which magnetic or magnetizable substances are added to an ore pulp, the whole being treated with an oil to cause the sulphide and the magnetic particles to adhere together. The oiled sulphides are then removed from the pulp by means of a magnet. Plants following this process have been erected and operated with some success. The process is not flotation, but since it uses oil and since the desired results are somewhat the same, it is of sufficient interest to be noted here.

James M. Hyde's Process.—In 1911, James M. Hyde brought out a flotation process in which a small amount of sulphuric acid, with or without the use of iron sulphate, was used to give the slimy portion of the ore a preliminary coagulation before bringing it to the flotation machine, and in which the sulphide, after agitation, was floated off rapidly and as completely as possible with a considerable overflow of freely flowing water. This thereby produces an impure concentrate that is retreated in a second machine, from which only high-grade concentrate is taken off, with no free-flowing water. The process has been adopted by the Butte & Superior Copper Co., of Butte, Mont. It has been in use at this mill for a number of years, and its principal importance is due to the fact that it constituted the first basis for litigation, the fact being that Minerals Sepa-

ration sued James M. Hyde for infringement of its patent in the application of flotation to the Butte & Superior ores. The Butte & Superior ores consist of blende widely disseminated and finely divided in a quartz gangue.

Callow Pneumatic Process.—A step of importance is J. M. Callow's American patent for his pneumatic machine, which was taken out in July, 1914. Some authorities claim that Callow's idea is the same as that appearing in T. J. Hoover's British patent No. 10929 of 1910, now the property of Minerals Separation. It has also been said that R. S. Towne used the same idea earlier than Mr. Callow in the form of a carborundum wheel, the central hole of which he plugged so that the wheel served as a porous bottom. Whatever the merits of the discussion may be, the fact is that Callow has succeeded in developing an important variation of the flotation process, and his means of applying the principles, whether they infringe anything else or not, is an effective one in many cases.

Wood's Surface Tension Machine.—While discussing the various flotation processes, it may be well to mention that of Wood, of Denver, who has, although not bringing forth any new process, developed a machine which takes great advantage of some of the surface-tension principles, and is so different from the rest as to be entitled to some mention. In Wood's method, the ore is crushed dry to pass a 30- or 40-mesh screen and is then fed in a thin stream from the vibrating plate onto the surface of water in a tank, to the surface of which a forward movement is given by small jets, also of water. By a combination of the capillary attraction and the pressure of a constant feed, the sulphides are caused to move forward as a definite elastic film on top of the water. This film of mineral passes over an endless canvas belt, which emerges from the tank at a particular angle, varied according to the kind of mineral to be saved. The belt, with its film of sulphides, passes over three rollers so that its motion is reversed and it strikes the water-level of the second tank, where it releases its valuable burden. Very little gangue in suspension comes over, and the water drains back into the main tank. Any submerged particles that have been accidentally wetted or so heavy that they have penetrated the surface film, pass to standard concentration tables, on which they are separated by gravity in the ordinary way. In the case of molybdenite and graphite, the film concentrate is still further cleaned

by being passed over a nearly vertical screen. Gangue in suspension passes through while the flat crystals of the valuable minerals slide over the screen, thus largely dewatering them. The flotation concentrate is collected and dried as usual. The Wood process has been used in many cases and at the present time seems to be largely successful.

Bradford's Preferential Separation.—Leslie Bradford, of Broken Hill, Australia, has patented a process of preferential flotation for the treatment of mixed sulphides, adding certain mineral salts, such as thiosulphates, to the water used in the flotation cells. The *Australian Statesman and Mining Standard*, of July 22, 1915, says that the distinguishing feature of the method consists in treating the ores or ore products in a medium which wets zinc sulphide and does not wet lead sulphide or pyrite and therefore leaves the last-named sulphides floatable while rendering the zinc sulphide temporarily immune to flotation. The lead sulphide and any pyrite are thus obtained first as float concentrates; thereafter the residues comprising gangue and zinc sulphide are treated for the separation of the zinc sulphide from the gangue.

This "wetting" effect, according to the patent specification, is procured by adding to an acid flotation solution a limited quantity of a substance which is decomposed by said solution, and produces in this decomposition a reducing gas which is more or less soluble in the solution. Notably thiosulphates, sulphites, and bisulphites are so decomposed in a solution of sulphuric acid with the production of such a gas (sulphur dioxide). With substantially equal result, sulphur dioxide may be introduced directly into the solution, either as gas brought into it in a current or added in solution. The like effect is obtained by means of another reducing gas, sulphureted hydrogen, which may be similarly introduced into the solution or may be produced in it by adding sulphides or polysulphides which the solution is capable of decomposing.

In practice a limited quantity of such a reducing agent is introduced into the solution, prior to flotation treatment of the ore therein, with the object of procuring this zinc "wetting" effect. The reducing agents preferred by the inventor are thiosulphates, sulphites, and bisulphites, as they are usually more effective than those belonging to the second group.

In the actual application of the invention to a Broken Hill

mixed-sulphide ore containing zinc and lead sulphides, the flotation operation is considered in a plant of the type described in two previous Commonwealth patents, using the flotation boxes set in series, and treating ore at the rate of 25 tons per hour. No serious difficulty is experienced in correctly proportioning the sodium thiosulphate (the blende "wetting" agent) so as to insure the rapid floating off of the galena substantially pure or as a high-grade concentrate rich in silver, and to leave the residue in a condition suitable for further flotation treatment for the recovery of the zinc sulphide therefrom.

The flotation treatment is preferably conducted according to the froth-flotation principle; the procuring of the effect aimed at is dependent upon the presence of a frothing agent in the flotation medium, only when a reducing gas is introduced into the medium; it is not dependent upon the presence of a frothing agent in the flotation medium when a reducing gas is generated in the flotation medium by reaction of a substance introduced into it. In some cases enhanced results are obtained by digesting the ore with sulphuric acid and the reducing agent in a thick pulp, prior to subjecting the same to a flotation operation.

In the foregoing discussion, practically all of the processes making use of real differences of principle have been mentioned. All of the machines invented for taking advantage of these principles have by no means been mentioned. There are too many of them, and new ones are constantly appearing. The merits of these machines, however, will have to be worked out in practice, and, aside from the ones already mentioned, there is nothing important enough to be noted in this section, although in the further description of the process herewith, many of them will be described.

Emphasis should be placed strongly on the fact that I have made no attempt to catalogue all patents that have to do with flotation. Only those that seem to be of practical importance in relation to subsequent commercial work have been mentioned and, no doubt, interesting ones have been omitted. No attempt has been made personally to search the patent records, but dependence has been placed on the literature, such as Hoover's original book and the files of the technical press.

CHAPTER III

THE THEORY OF FLOTATION

The most peculiar fact about the flotation process of ore concentration is that in spite of its rapid development and wide application to metallurgy, the practical results obtained have not been built up as a result of the application of a definite theory, but have rather disregarded basic reasons and grown upon the practical physical facts derived from experimentation. The theory of the process, up to this time, has been uncertain and none of the tentative theories so far proposed has been sufficiently elaborated and discussed to prove entirely satisfactory. The surface-tension theory, undoubtedly a primary basis of some flotation phenomena, is believed also to account for the production of froth by the use of oil through the fact that oil reduces surface tension of liquids and allows bubbles of a more permanent character to be formed. While partly explaining the phenomena, this one physical fact will not account for all the results that come through all kinds of flotation processes.

Surface tension has been elaborated in the discussion of theories by Hoover* to describe its effect and its influence upon

*"Concentrating Ores by Flotation," First Edition. Page 49.

the flotation process. He points out that all masses of substances, whether solid, liquid, and probably gases also, have acting at their surface a force parallel to the surface that is the resultant of molecular activity within the body of the substance, and that tends to resist rupture of the surface. In the case of a liquid this force is known as surface tension and in the case of some liquids the force has been measured. In Hoover's description, it is noted that for the surface of plain water in contact with air, the magnitude of this force has been determined to be 81 dynes per linear centimeter.

Discussion of Surface Tension.—It may be better, however, before going into a detailed examination of surface tension, to examine the explanation of that force.

W. R. Ingalls¹ says that a body of water, or of any liquid,

¹ *Eng. and Min. Journal*, Oct. 26, 1907. Page 765.

has greater cohesion in its free surface, *i.e.* the surface exposed to the air, than anywhere else in its interior, because, whereas the particles in the interior are mutually attracted by adjacent particles in all directions, those at the surface have no attraction from the outside—from above, let us say—to counteract the pull from those in the interior. This condition may be illustrated in diagrams by the accompanying sketch adapted from Hoover's book. The molecules of the liquid are imagined to be of sufficient size to be represented on paper, and in the drawing each circle represents a molecule. I quote this explanation directly from Hoover:

The molecular attraction of each molecule can be resolved into resultants for purposes of illustration, the resultants being placed as shown by arrows in the figures. To make the assumption complete we must imagine two resultants acting at the center of each circle, at right angles to the plane of the drawing. The molecule at *A*, for instance, attracts and is attracted by adjoining molecules by resultant forces that are equal, and the molecule is free to move in all directions. The molecule at *B*, however, having no molecule above it to attract and be attracted, is in a somewhat different state. The molecule at *B* has its upper one of the fixed resultant forces uncompensated by a molecule lying above it, so we can assume that this upper resultant is

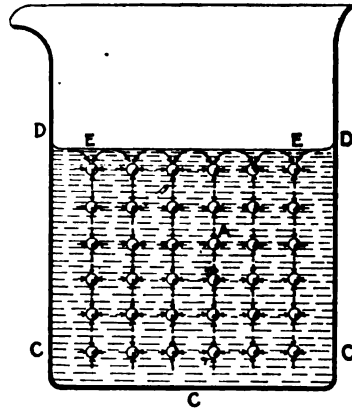


FIG. 3.—An illustration of surface tension.

compensated between the molecules on any side of *B*. This gives the topmost row of molecules a stronger attraction for each other than the molecules adjacent to *A* have for one another. The attraction of the molecules within the liquid, as for instance those at and adjacent to *A*, is known as the force of cohesion. If this force of cohesion is resolved into the six resultants, as above described, and its magnitude is called 6 units, then the magnitude of the molecular attraction acting on a molecule in the top layer, as at *B*, will be six units just the same, but the molecule will not be as free to move in every direction as the molecule at *A*. If the stress between any two molecules in the horizontal row containing *A* is 2, then the stress between any two molecules in the horizontal row containing *B* will be $2\frac{1}{2}$.

Ingalls goes on to say in explanation: "The effect of this is to reduce the mobility of the particles on the surface. The surface, as it were, is stretched by an elastic skin, the effect being the same as if the surface layer exerted a pressure upon the interior. The production of this surface tension represents energy, *i.e.*, molecular energy, which may be measured."

As has been quoted above, this surface tension has been determined for water to be 81 dynes per linear centimeter, a force that resists penetration from within or without the liquid.

Surface Tension in Everyday Life.—The effects of surface tension are apparent in a great many of the ordinary appearances of modern life. So common are they that no attention is usually paid even to their most wonderful manifestations. Some of these ordinary occurrences are described by T. A. Rickard, in a paper he presented at the March, 1916, meeting of the Canadian Mining Institute, at Ottawa. Descriptions of the force and its manifestations have been voluminously described by various writers in the technical press of the last year or two, but it may be assumed that to the reader deeply interested in the theory of flotation, these illustrations are hardly necessary, so they will be passed over at the present time and more attention will be given to the purely practical phases of the subject so far as we know them. The many articles already referred to form a sufficient and satisfactory basis for any one who cares to, or needs an elementary explanation of the phenomena. Will H. Coghill¹ presents an argument disagreeing with the previously made explanations of surface tension. He refers to the book on "Surface Tension and Surface Energy," by Willows and Hatschek which shows how the elastic-film analogy of the study of froths may get the student into trouble if not handled with care. He draws attention to the theory of surface tension, which is explained, or rather illustrated, by the analogy of the stretched rubber film. Analogies of the kind are liable to become destroyed if pressed too far, although they are useful up to a certain point. This one, says Coghill, is no exception. In the case of India rubber it is obvious that a given weight can only stretch this to a definite extent. To further enlarge the rubber film, an additional weight would be required, while with the liquid film this is not true. As to the breaking of the film, this depends, according to Coghill, not upon the surface tension, but whether

¹ *Min. and Sci. Press*, Feb. 29, 1916.

or not there is enough liquid to supply the added area for enlarged bubbles. The difference between a non-aqueous substance and a liquid film is that in stretching the former the molecules are distorted or separated, while in stretching a liquid film the molecules come from within the liquid to occupy the new area. According to Devaux, the surface-tension phenomena disappear as soon as there is no more liquid to come from within. The same laws apply to surfaces that are allowed to contract. The rubber has a constantly decreasing force of contraction as it approaches its original dimensions, while a liquid film always tends to contract with the same force independently of its size. It is a common practice in demonstrating physical principles to omit certain qualifying conditions until the main features are outlined. This method must be pursued here. The qualifying statement is that in the case of a contaminated liquid the film may not contract with a force independently of its size; that is, after learning to look upon surface tension as a constant force, we must now view it as a variable one. Coghill apparently believes that variable surface tension is to be accounted of importance in froth formation, rather than uniform reduction of it.

Surface-Tension Processes.—The descriptions given here of various interpretations of the phenomena of surface tension are presented with a view of pointing out the trend of opinion on the subject. Undeniably for such processes as the Macquisten and Wood, where minerals are floated upon water surfaces without the addition of oil or other chemicals, the actual surface energy is probably the principal force concerned. Here the mineral particles are deposited with care upon the water surface, and due to the surface tension and the property of minerals to present a contact angle of a certain degree (which is the same as saying they are non-wet), these minerals float. The siliceous gangue does not float, a fact that is due to the inherent tendency of such particles to possess an angle of contact of a decidedly less degree, as a result of which the particle is wetted, the surface film is ruptured, and the particle sinks.

When one speaks of flotation by means of the air-froth method as depending upon surface tension the statement is, perhaps, accurate, but it is not always clear whether the one making the explanation in such terms is clearly aware of the force of his statement. The forming of a froth has nothing necessarily to do with floating minerals. One experimenter reports that he can

form a very voluminous froth with saponin, but gets no metallurgical separation by it. The minerals do not adhere to this froth, but will sink through it just as readily as siliceous particles will. Saponin "kills" the froth made by oils. The actual flotation by froth involves the same force, and when all points are reviewed carefully, it will be seen that the application is merely another phase of the same forces, applied in a slightly different way.

Unexplained Forces.—Froth flotation depends in part upon adhesion, and although it may be proved that this statement is technically accurate, it is very difficult to go back of it and explain what adhesion is. The manifestations of adhesive force are as clear and evident as those of the force of gravity, and just as unexplained. When an oil is added to a mineral-bearing pulp, there is a certain definite tendency for the oil to form a film upon the mineral particles, and there is a certain definite tendency for them to float upon the surface of the bubbles. To these tendencies the metallurgical world is indebted for the flotation process of ore concentration. These forces have not been really explained, and no matter how many words may be expended upon an effort to clear the situation, the facts must simply be taken as they exist, in the present state of our knowledge.

Electrical Surface Charges.—A word may be said here in reference to the often repeated statement that electrical charges upon the surfaces of the solids having something to do with the phenomena brought about by flotation conditions. It has been shown that there is a connection between the charge on a particle and its contact angle, thus affecting the degree of wetness, but circumstances are such that explanations can be better made on some other basis at this time. It is true that particles of any size may, and probably do, carry electrical charges of some kind, since in most cases such charges may be measured under proper conditions, and these surface charges in the absence of other films may have much to do with the physical action of the particles, but conditions in ordinary work are not such that this force can be considered of commercial importance.

The Matter of Colloids.—The mention of colloids brings us face to face with another problem, the elucidation of which has much to do with a proper understanding of flotation theory.

The metallurgical world is indebted to E. E. Free,¹ who has given us a view of what colloids really are and what can be done with them. Quoting from Free's article, the following is of interest:

. . . we have developed a conception of colloids that regards them simply as suspensions of very fine particles. The misconception of colloids as specific gelatinous substances possessing unusual quasi-chemical affinities we have criticised because it rests upon unsupported hypotheses and because it is unnecessary and confuses a matter not otherwise obscure. We have made no sharp distinction between suspensions and colloidal solutions. The emphasis, indeed, is quite the reverse, and is on the unity of this theory rather than on any possible severalty. As always, our insistence is upon properties rather than substances. Certain colloidal properties, among which we have emphasized slowness of settling and imperviousness of the settled mass, characterize suspensions of finer grain. These we call colloids. The distinction is purely relative. We mean that these suspensions are more colloidal than those usually encountered.

. . . Several other properties are of interest and deserve brief review. Most important scientifically are the similarities to, and differences from, the true or ordinary solution, the particles of which are supposed to be of molecular or ionic dimensions. The typical colloidal solution resembles a true solution in being permanent so long as conditions remain unchanged. That is, the particles remain in suspension and the colloidal solution retains an unchanged chemical composition. The colloids differ in failure to show a true and constant solubility; an optical heterogeneity shown by translucence or turbidity; the causing of no change or very small change in the freezing point and boiling point of the solvent; the production of no osmotic pressure; slow diffusion and failure to dialyze or pass through a parchment membrane. The causes of all these differences will be evident on consideration of the difference in particle size which distinguishes the colloids from the true solution. Details need not be pursued further than to say that the differences, as before, are of degree only. Colloids do not fail entirely to show the typical solution properties, but they do show them to so slight a degree as to escape all but the most painstaking and exact researches.

Of other properties of colloids, flocculation and absorption are mentioned. Another of much technical importance is the tendency of the colloidal particles to wander in the electric field and accumulate at one or the other of the poles. This is distinguished from ordinary electrolysis by the fact that the passage of current is not essential.

¹ *Eng. and Min. Journal*, Feb. 5, 1916, and subsequent articles.

The matter is one of electrostatics and occurs in non-conducting media. It forms the basis of the well-known Cottrell process for the collection of acid and smelter fume, the removal of dust from stack gases, the separation of oil globules from condenser water, and the like.

In Free's definition of colloids he has, by inference at least, limited it to solids. This is legitimate for our purpose, since it is with solids—ores—that we are concerned, but it is not general. W. D. Bancroft¹ says that "we call a phase colloidal when it is sufficiently subdivided, not limiting ourselves definitely as to what degree of subdivision. Colloid chemistry is therefore the chemistry of bubbles, drops, grains, films, and filaments."

Colloid solutions can be prepared by precipitating the substance from vapor or true solution in a finely divided form, or by disintegrating a mass of the substance in some way, mechanically, chemically, etc. *Stable* colloidal solution cannot be produced by subdivision alone. To keep particles from coalescing there must be either a material film, such as gelatine, or an electrical charge due to an absorbed ion; except under special conditions.

Having thus acquired a conception of what colloids are, we can now attack the problem of colloids in flotation. The colloid condition is not favorable to commercial flotation, a fact that experimenters who have been trying to float precious-metal slimes have come into contact with rather forcibly. Matter in solution cannot be subjected to flotation (at least, within the limits of the practice of the subject under discussion), and it might seem, too, that colloids would also be outside flotation influence, but W. D. Bancroft² has shown that under proper circumstances some colloids may be easily floated.

"Isobutyl alcohol was added to a colloidal gold solution obtained by reducing gold chloride with carbon monoxide. When the two liquids were shaken the gold forms a thin film at the interface. This film is violet blue to blue green by transmitted light and golden by reflected light. A thin water film forms between the isobutyl alcohol and the glass, and the gold concentrates in the dineric interface thus formed, making the alcohol appear uniformly gold-plated. With ether the gold film rises high above the level of the two liquids. With carbon bisulphide the adherent film of gold appears blue. When the carbon bi-

¹ *Jour. Phys. Chem.*, 18, 549 (1914).

² *Met. and Chem. Eng.*, June 1, 1916.

sulphide is broken into drops by shaking, each drop appears blue. When a blue gold was obtained by reducing gold chloride with phosphorus dissolved in ether, the gold went into the dineric interface. When a brownish-red gold was obtained in this way, it remained in the water phase and showed no tendency to pass into the interface. This difference is undoubtedly due to an adsorption of something at the surface of the gold, because Reinders found that 0.005% gum arabic prevents colloidal gold from passing into the ether-water interface. With carbon tetrachloride, carbon bisulphide, or benzene, the gold goes into the interface as before, but the gum arabic prevents its changing from red to blue.

"Colloidal arsenic sulphide goes into the dineric interface with amyl alcohol or isobutyl alcohol, but stays in the water phase when carbon tetrachloride, benzene, or ether is the second liquid. India ink goes completely into the interface with amyl alcohol, carbon tetrachloride, or benzene; it goes partly into the interface with isobutyl alcohol, and stays entirely in the water phase when ether is the second liquid."

The Matter of Emulsions.—Now, building upon this foundation, attention is called to the inaccurate use of a word that has appeared so often in recent contributions to the science of flotation. Writers on the subject frequently refer to the combination of pulp, oil, water, acid, or whatever other chemical substance is added to it, as an emulsion. Going back to the definition of an emulsion, we find that it is a colloid suspension of one liquid in another. For example, the oil in a mixture may be so finely subdivided as to approach the colloid state and simulate a true emulsion. If, as has been said, the colloid condition is antagonistic to flotation success, then this pulp-oil combination is not a true emulsion, since we clearly do get flotation results from it. The latter statement is true—it is not an emulsion.

The physical evidence of the non-colloid condition of the pulp subjected to flotation conditions is that of a mixture and not an emulsion. In undertaking a careful study of these conditions, we are again indebted to Free, who, in the series of articles already mentioned, has given a clear exposition of these conditions under the subject of "Sedimentation and Flocculation."¹ Here he has pointed out the factors that tend toward floccula-

¹ *Eng. and Min. Journal*, March 4, 1916.

tion and those tending toward deflocculation. Among the flocculating agents is mentioned sulphuric acid. Now, since a true emulsion is a colloid condition, it is also a deflocculated condition. A deflocculated condition does not commonly exist in the presence of sulphuric acid. The deflocculated, or colloid, material remains as a more or less permanent suspension, and flotation conditions have no effect on it. We have therefore the interlaced and interdependent facts that a flocculated condition is desirable for flotation, while deflocculation is a requirement of emulsion. Sulphuric acid is usually a flocculating agent; therefore, emulsions are not to be expected when sulphuric acid is present. It is extremely probable that the sulphuric acid, a flocculating agent, is mainly used to prevent the colloid or emulsion condition, although a great many other reasons have been given for its use. The undesirability of the emulsion condition of oil in flotation and the function of sulphuric acid in preventing emulsification was first pointed out in the work of G. D. Van Arsdale. Among the other reasons given for the employment of sulphuric acid may be mentioned that of cleaning of the surfaces to permit the adhesion of the oil film to a mineral or metallic surface. Apparently, however, the amount of acid commonly used is too small to have any active chemical effect in such a bulk of water and solids, but is in quite sufficient quantity to act as a flocculating agent. Hence, in speaking of the pulps used in flotation, they should be referred to as mixtures and not as emulsions.

There are of course differences in the character of emulsions, and these differences depend upon the character of the liquid that approaches the colloidal state. Many emulsions are such that when set aside the finely divided particles of oil coalesce and separate from the solution mass, while others are of such character that the colloidal condition is maintained and the fine particles do not coalesce or separate even upon long standing. It may be shown that oils in the colloidal state, or emulsified, do not promote flotation. Apparently their effect upon the actual surface tension of the liquid is the same as when in any other condition, but when in the colloidal state these oils develop physical characteristics that tend to force them to maintain that condition and therefore the oil particles remain suspended separately and refuse to oil the mineral particles. Apparently the intersurface forces set up preclude the selective action of the

oil upon the minerals, and therefore no flotation results. However, if a truly emulsified oil in pulp is treated with sulphuric acid, tending to flocculate the oil, the selective quality of the oil-mineral combination comes into action again, and flocculation conditions are reestablished.

The Degree of Flocculation.—It may be said, quoting again from Free,¹ that the degree of flocculation is extremely sensitive to surrounding conditions; so sensitive indeed that changes of flocculation in the true colloidal solutions are among the most delicate of analytical reactions. These changes and similar ones in the coarse suspensions may be caused by many kinds of influences. Temperature, light, the radiation from radium, electrostatic charges, and many other variations in energy relations all appear to have perceptible effect. However, these effects of direct energy changes are far less important practically than the effect of added substances, and it is these latter alone that require consideration in a practical examination of the subject. A large amount of observation and incidental experiment, going back for many centuries, has sufficed to establish three general classes of substances which affect this degree of flocculation, or, as it is generally conceived, the rate of settling of suspensions. First are the neutral, inorganic salts and the inorganic acids, all of which increase the degree of flocculation and promote settling. Second are the alkalis, which (in certain concentration) have the reverse effect, decrease the degree of flocculation, hinder settling, and are known as deflocculators. Third are the organic colloids of the liquid-particle type (emulsoids) such as gelatin, albumen, and the like. The effect of these is somewhat complex, but in general they also decrease the degree of flocculation, or at least prevent its increase.

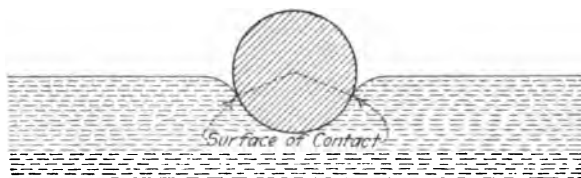
A careful explanation of how flotation takes place has been worked out by G. D. Van Arsedale.² He prefaces his argument with the statement that it is easier to tell how flotation works than why, and consequently he leaves out any attempt at basic reasoning, but does show the actual uses of the various processes employed. In the pages immediately subsequent I shall follow Mr. Van Arsedale's line of argument in the main.

¹ "Sedimentation and Flocculation," *Eng. and Min. Journal*, March 4, 1916.

² "How Flotation Works," *Eng. and Min. Journal*, May 13, 1916.

Mutual Surface Action.—It is not so easy to give an actual definition of flotation, but it is possible to say that it is concerned with the mutual action of the surface of solids, liquids, and gases, and that the surfaces of various solid particles behave differently with respect to the surfaces of liquids and gases. It is possible, using this definition, to classify practically all the known methods of flotation. The first class makes use of the surface of one liquid and is exemplified by the Wood and Macquisten systems. The second class employs the surface of one liquid and of gas bubbles, and includes the Potter-Delprat method. The third class utilizes the surface of two immiscible liquids, and is exemplified by the first Elmore process. Class four is the utilization of the surface of two immiscible liquids with the addition of gas bubbles; this class includes the modern oil-flotation methods.

The froth, which is one of the most prominent features of ordinary flotation methods, is a collection of bubbles, and a bub-



$$\text{Sphere, } P = \frac{T}{R}, \text{ Cylinder, } P = \frac{2T}{R}.$$

FIG. 4.—The surface contact diagram.

ble consists of a liquid surface in contact with gas, which in some cases it entirely incloses. This definition connects the basic theory of froths with the process used in class one, since the conditions are the same—a liquid surface in contact with a gas. In other words the froth utilizes ordinary surface-tension forces.

A particle heavier than water can only float upon a watery surface because of the effect of surface tension. There is a definite relation between surface-tension factors and the maximum weight that can be supported on water surface, and this has often been quoted in the statement that one square centimeter will support about 81 mg. As a matter of fact, Mr. Van Arsdale shows that a square centimeter will support much more than this weight. The desired relation can be derived from the surface-tension formulas for the difference in pressure between two kinds of a curved surface and these formulas are shown in

the accompanying figure. By their use the maximum size of cylinder and sphere that will float can be derived, and as a matter of fact clean copper wires of a size very close to that calculated can be floated, indicating the correctness of the hypothesis. The length of the wire should make no difference according to the theory, and in fact, a long wire floats just as well as a short one.

The question of influence of the shape of the particle can be answered by referring to the same formula. It can be seen from them that with equal radii and surface of contact the specific gravity of a maximum-sized floating sphere may be twice that of a floating cylinder, showing that the shape of the particle is an important consideration.

Effect of Surface Films.—The other conditions to be taken into account are the presence or absence of surface films on particles, the kind of films, and the resulting contact angle produced by them. It is perfectly evident to all acquainted with flotation that films of various kinds on particles may prevent or produce flotation. A clean copper wire floats. The same wire having an oxide film sinks promptly, while the same wire, again, with a sulphide film floats easily. Mr. Van Arsdale points out that the reason for this difference in floating ability is the character of the contact angle produced by the film, but W. D. Bancroft is inclined to question the possibility of measuring a contact angle with any degree of accuracy and therefore is not sure that a specific contact angle for a substance exists, but is variable widely due to many conditions. As a matter of fact, however, these films do have the effect of promoting or preventing flotation; and such films may be extended to include charges of electricity, theoretically, at least. In continuation of the contact-angle theory, Valentine has set forth, theoretically and by practical tests, that one of the flotation factors is the contact angle, which must be of a certain degree to make floating possible. The film, as has been said, may be solid, liquid, gaseous, or even an electric charge. The use of oil in flotation practice exemplifies the liquid film, which will produce as definite results as the solid film. Particles of malachite converted to sulphide by the action of H_2S gas, if dropped on the water surface, float well because their contact angle is due to their having a strongly adhering surface film of gas.

Up to this point, the theory has been extended to several con-

ditions of flotation, namely: surface tension, size and specific gravity of particles, shapes of particles, and films and their resulting contact angles or whatever name may be applied to the effect produced by films upon particles which promote or prevent flotation. Any particle heavier than water may be made to float provided it is small enough, of proper shape with respect to size and weight, and has a surface, the resultant of which is a contact angle that permits flotation.

Surface Films and Gas Bubbles.—It is practically impossible to produce a surface free from a film of any kind, but it seems likely that for all such surfaces the contact angle with water would be zero, or it may be otherwise stated in that a perfectly clean particle will not float. It is of importance to know why gas bubbles attach themselves to mineral particles and not to gangue particles, and whether there is any particular virtue in the way gas bubbles are generated by the action of acid on carbonates of the gangue in such systems as the Potter-Delprat. If some

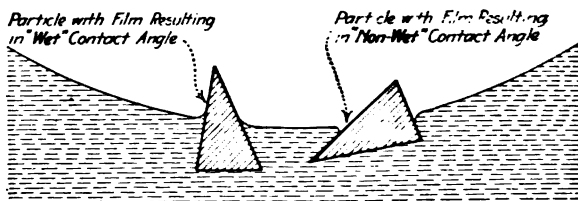


FIG. 5.—Magnified section of a bubble.

mineral particles whose contact with water renders them non-wet are put into the bottom of a vessel containing pure water and a bubble of air is brought into contact with these particles, they will become attached and the particles will be carried up to the surface with the bubble. This goes to show that it is not necessary to form the bubble in solution, nor is it necessary to have nascent bubbles or anything of the sort. The mere attachment of the mineral to the bubble, or the simple contact of gas and liquid surfaces is all that is necessary. In explaining why the bubble attaches itself to the mineral and not to the gangue, the accompanying sketch represents a largely magnified diagram of a bubble of air or gas in contact with two particles, one of the particles having assumed a contact angle that would enable it to float on a watery surface, the other particle having a contact angle that would produce immediate sinking on a water surface.

Considering this diagram, it will be seen that the bubble is really only the surface of water in contact with air or gas, and that the particle floats on this surface for exactly the same reason given in the first class of methods. The fact that the system of bubble and particle is small and movable does not influence the surface tension and other factors that govern this kind of flotation. There is no attraction between bubbles and particles at a distance, but as soon as this surface comes in contact with the non-wet particle, it attaches and floats on the surface exactly as on a larger water surface.

There are some other methods whose action bears directly on modern processes, and among them may be mentioned the bulk-oil or first Elmore process. This is explained in the fact that certain kinds of particles enter and remain in the oil, others remain on the surface separating the oil and the water, and others again enter and remain in the water. All three can be explained on the basis of surface tension together with other factors already mentioned, and Reinders has proved this both theoretically and practically.

Reinders's Statement of Inequalities.—"If we represent the interfacial tension between the solid and water as $T(s, w)$, the tension between water and oil as $T(w, o)$, and the tension between the solid and oil as $T(s, o)$, Reinders stated that:

"If $T(s, o) > T(w, o) + T(s, w)$ the solid powder will remain suspended in the water.

"If $T(s, w) > T(w, o) + T(s, o)$ the solid will leave the water and go into the oil.

"If $T(w, o) > T(s, w) + T(s, o)$, or if none of the three tensions is greater than the sum of the other two, the solid particles will collect at the boundary between the oil and water."¹

These expressions will also hold true for a particle coming into contact with a film of oil instead of a layer or globule of oil, except that in this case the particle becomes coated with a film of oil or does not become so coated, instead of entering or not entering a layer of oil. It should be emphasized that the reasons for the action as shown by these illustrations are precisely the same as those given for the first two classes of methods—namely, size and shape of particles and the kind of film on such particle or absence of film, which produce by their presence or absence the resulting surface tension and contact angles.

¹ From article by O. C. Ralston, *Min. and Sci. Press*, Oct. 23, 1915.

Action of Oil.—The action of oil may be illustrated by experiment. A globule of oil is suspended in a mixture of equal specific gravity. A silicate particle coming in contact with this oil will not be wet by it, as may be seen by trying to stir about such a bubble of oil with a glass rod, which represents the silicate particle. If the same operation is carried out with a copper wire having an oxide film, the same result will be obtained, but if it is tried with a copper wire having a sulphide film it will readily be seen that the oil will definitely adhere to the wire and it is only with some difficulty that it can be disengaged, and when it is disengaged it will still leave a film of oil upon the wire. The reasons have been set forth in the inequalities of Reinders already mentioned.

What happens when a particle meets oil in the form of emulsion is not altogether clear, but it is very likely that there will be no attachment of oil to mineral unless some factors are introduced that will break up or destroy the emulsified condition, thus bringing out a fact that has already been referred to. Since sulphuric acid is largely used in flotation, it is altogether possible that this is one of its principal uses. Further proof is found in the fact that substances that promote emulsification are disastrous to flotation. The accompanying table shows the effect of various agents upon emulsion.

EMULSION-MODIFYING AGENTS

Sulphuric Acid.....	0.244%	entirely prevents emulsification
Cresylic Acid.....	0.008%	entirely prevents emulsification
Acetic Acid.....	0.014%	entirely prevents emulsification
Citric Acid.....	0.066%	entirely prevents emulsification
Alum. Sulphate.....	0.06%	prevented emulsification
Copper Sulphate.....	0.30%	prevented emulsification
Ferrous Sulphate.....		No action
Caustic Soda.....	0.30%	No action
Sodium Chloride.....		Stabilizes and increases emulsification

As a means or mechanism for separation of mineral and gangue particles, the froth is, of course, simply a very largely extended surface as compared with the similar surfaces of water, and this extension of surface will be more or less proportional to the size of the bubble. If a space 6 in. deep above a water surface 1 ft. square is covered with bubbles, the amount of increased surface

may be taken as roughly equal to the figures shown in the table given herewith.

SURFACE AREA OF FROTHS	
1 sq. ft. of froth, 6 in. deep	
Average diameter of bubbles	Surface area at least
1 in.	18 sq. ft.
$\frac{1}{2}$ in.	37 sq. ft.
$\frac{1}{4}$ in.	75 sq. ft.
$\frac{1}{8}$ in.	150 sq. ft.

Qualities of Flotation Oils.—The question of oils comes in next and the object is to find out what use is to be specified for oils. Oils are, of course, used for two main purposes—first to provide a film around the particle and second as a means of making a foam or froth. We can then divide the oils into “oilers” and “foamers,” and in practice a mixture representative of both classes is desirable. Of course, for oilers an ordinary cheap oil, possibly petroleum, can be used provided it supplies the necessary film upon the mineral particle. In the second class of oils is needed one which reduces surface tension most. Professor Bancroft calls attention to the fact that an absolutely pure oil will not produce froth, and that the frothing quality of an oil is due to colloids in the oil. It may also be added to this statement that fine enough particles in the pulp will have the same effect, so that we may really have a froth made up of an oil having no colloids in it originally, the colloids’ effect being supplied by the particles.

In continuation, Mr. Van Arsdale says that the result of the addition of the second class of oils is that a minute amount of any of them, such as crude pine oil, cresylic acid, etc., added to a surface of water upon which a film of oil of the first class is spread, will result in an immediate and striking retraction of the film. The oil then takes on, in most cases, a circular, globular form, with definite contact angles. This shows an important function of the addition of this minute amount of, for example, crude pine oil, is to change the surface tension and contact angle of the other oil or the particle coated by it, so as to enable it to float, which it otherwise would not do. Heat may

apparently act in the same way, for if some oil is dropped upon the surface of hot water it does not film out but assumes a globular form.

If this reasoning is correct, it seems that instead of adding oils in a mixture it will be found more rational to separate them and add the proper amount of the first class to mixing apparatus or mill machinery, and then after having coated the particles, to add the second class of oil to the flotation apparatus. This ought to give better practical results, since it is evident that during the first operation it is necessary to have the oil in such a condition that its filming-out properties will be at a maximum, which will not be the case with a mixture of both classes of oil as ordinarily used. Along these lines it is possible to imagine that in some cases it may be possible to omit the frothing oil altogether, since, if with Professor Bancroft's ideas correct, the frothing is not due to the oil at all but to colloids that might possibly be supplied in some other way. As a matter of fact, as pointed out by Mr. Van Arsdale, it is not impossible that we may finally have a successful flotation without any froth, with its consequent power consumption, or even without the use of any oils.

The Rôle of Films.—Some further discussion on films and their action has been presented by Will H. Coghill, in the *Engineering and Mining Journal*, July 14, 1917. He quotes from Milliken a law of transmission of pressure by liquids. It is that the molecular pressure existing because of adhesion at a point in the liquid close to the limit of contact is transmitted undiminished in the direction parallel to the surface of the solid, and therefore constitutes a force pushing out the molecules. The only opposing force acting to prevent the limiting molecules from moving along the surface is the vertical component of the attraction exerted upon these molecules by such force of the liquid as lies within the sphere of influence about the center. Hence, unless the ratio of cohesion to the adhesion exceeds a certain limit, a thin film of the liquid must spread out indefinitely over the surface of the solid. This conclusion is not surprising, since it means simply that a body that attracts a liquid firmly enough will draw every particle of it as near as possible to itself. It results, clearly, that the more the molecular pressure because of adhesion exceeds the cohesion of the liquid itself, the greater will be the spreading power.

It is necessary that a solid, in order to promote great adhesive power, must have a perfectly clean surface. There are any number of illustrations of this in the ordinary life of the metallurgist, among which may be mentioned the difficulty of coating a copper plate with mercury and the difficulty of amalgamating gold. In each case this is because the solid is not easy to clean absolutely. The process of coating iron with zinc is another illustration of the same thing. Unless the iron be perfectly clean, a continuous and perfect coating of zinc is not obtained. The impurities, apparently, lack the adhesive power that the iron possesses, and therefore must be removed.

Mr. Coghill further points out that there may be difficulty in recognizing the processes of amalgamating and galvanizing as physical phenomena, more readily accepting the belief that they are due to chemical forces. This difficulty is cleared up by referring to the observations of Langmuir, in an article in the *Journal of the American Chemical Society*, Nov. 1916, page 2225, in which occurs the following:

"In gases there is usually no uncertainty as to the size of the molecules, but in liquids and solids no really satisfactory method has been found for determining molecular weight. So long as we cannot definitely determine the molecular weight, it thus remains impossible to distinguish carefully between chemical and physical phenomena. Nevertheless, much discussion has arisen of late years over such questions as whether absorption and surface tension are chemical or physical phenomena. The old consensus of opinion seems to be that these are both physical phenomena."

In pointing out that there is no justification for the distinction between chemical and physical forces, Langmuir further says: "Cohesion, absorption and surface tension are all manifestations of forces similar in nature to those acting between the atoms of solid bodies. It is therefore advantageous to look upon these forces as the direct results of chemical affinity.

It is readily seen then, that if it is desired to spread a liquid upon a solid, one of two things, or both, may be done. The adhesive power of the solids may be increased, or the cohesive power of the liquid may be diminished. Cleaning the surface of the solid and polishing it, may increase its adhesive power. Liquids may be modified so as to reduce their cohesive power by making certain additions to them. For example, though

plain water does not spread very well upon coal, it may be assisted by adding soap to the water. Acids also reduce the surface tension of water and reduce its cohesion.

It is pointed out, also, that this principle has been incorporated into the science of detergents, resulting in the factor that the power soap solutions have of wetting oily substances is due to the soap itself. It is due to the strong adhesion of the soap to the oil and the low cohesion of the soap to itself. It is seen that in addition to wetting and thus removing oil, a soap solution has the power of wetting other solids and so loosening them that they will slip from the material that is being washed.

The same principle has been employed, as is pointed out by Coghill, in the preparation of liquids for spraying trees and shrubs. In this case, of course, one of the prime requisites is to have a liquid containing such poisonous properties as will destroy the life that is inimical to the plant. Even though the liquid contains the proper poison, however, it is of no use unless it also has the property of spreading over and wetting the plant itself. The great importance of the spray fluid having a high wetting power is gradually becoming more and more recognized.

Inhibitory Films.—Coghill points out that the problem becomes more difficult when there are two immiscible liquids, and it is to be determined which will spread upon a given solid. R. C. Canby says that the sulphide surfaces are so delicate that even the mere drying of wet surfaces may have an effect that is sufficient to prevent successful flotations. This trouble is due to the films that are formed upon the surfaces when dried, since there is a group of solids on which water inhibits the spreading of oil, an intermediate group which is uncertain, and a third group upon which oil inhibits the spreading of water.

The first group is probably the most familiar. As an example, water inhibits the spreading of oil on quartz. Water in the form of extremely small percentages of moisture will prevent the spreading of oils on glass an effect that can be overcome by heating the glass so as to dry off the water, and then putting the oil upon it.

If perfectly dry materials of some kinds, such as clay, are coated first with oil, it is difficult to wet them in water. This obstacle has been encountered by the potters, who have found that oily clay grains prevent the display of plastic properties.

A further argument of Coghill in this article is quoted directly. "Again, we can apply well-known principles and show that the spreading of oil at the water-sulphide interface is far from paradoxical. It will be recalled that our working definition of surface tension incor-

porated the principle that the combined effect (from the viewpoint of energy expended) of a molecule moving about within a liquid was zero. Now imagine a drop of oil in water. It can move about with perfect freedom because the retarding force caused by pulling the molecules apart on the advance side equals the accelerating force of reunion at the rear. But assume that this drop of oil has removed to the water-sulphide interface. Here it receives a net accelerating force along the solid surface because the energy of the water molecules closing behind it is greater than that required to separate the water layer from the solid. Because of this unbalanced force the oil droplet will spread between the water and the sulphide, just as honey putty between the fingers when squeezed in the hand. This is, of course, in opposition to the law of minimum potential energy of surfaces, for it is obvious that the droplet experiences an enormous increase in surface, and therefore surface energy, but it must be borne in mind that liquids only tend to reduce surface to a minimum. There are many ways in which this tendency meets opposing and even controlling forces.

The oil therefore spreads between the two phases. It spreads on either water or sulphide at the air interface, therefore it seems natural rather than paradoxical for it to spread at the water-sulphide interface, if the water has not already spread. Having once spread, it forms an inhibitory film that checks other liquid from coming within the range of molecular activity of the sulphide. When the sulphide has depressions filled with entrapped air, as doubtless they have, the oil does not have to break the adhering band at these spots, but spreads freely on both the air-sulphide and air-water interfaces.

Oil cannot spread upon quartz in water because water has spread upon it and formed an inhibitory film. It might be urged that oil will spread on water-filmed quartz in the air, but such an oil film could not be expected to remain unaltered when submerged, because it is well known that a drop of oil between layers of water draws up into a sphere. If we increase the surface tension of water so that the cohesion is sufficiently great to prevent water from spreading on the quartz, then oil will doubtless spread at the quartz-water interface. If we add pine oil to our water with the consequent reduction in cohesion, so that the water spreads on the sulphides, it will be useless to add tar. But it is not done in that way. The tar is added first, and after it has formed an inhibitory film over the sulphide, we add the pine oil. The pine oil doubtless assists by producing a contaminated water that is more effective than pure water in spreading over quartz, but if it were of such a nature that it made the cohesion so small that the liquid spread on tar we would be defeated in our tarring; we would, in reality, convert the flotation machine into a washing machine."

If sulphides become refractory on exposure to the air, it is

probably due to the formation of a film with sufficient adhesive power toward water to make it spread. It seems consistent, therefore, in this case, to undertake the tarring in the presence of strong brine and then, after the removal of the brine, proceed in the ordinary. There is the possibility, of course, that the tar would peel off prematurely. A tar with a very low surface tension should be chosen.

Explanation of Froth Formation.—A recent explanation of the principles of flotation in general and of froth formation in particular, is contained in a paper prepared by Arthur F. Taggart, Assistant Professor of Mining Engineering at the Sheffield Scientific School, Yale University, in the early part of 1918, and published by the Franklin Institute, of Philadelphia. Taggart points out that the two essentials to successful froth flotation are, first, the selection of the valuable metallic material to the exclusion of gangue, and, second, the formation of a froth of sufficient stability to allow its separation from the bulk of the pulp. If particles of sulphide and gangue, immersed in water, are touched with oil, the oil spreads out on the sulphide surface and assumes a globular form on the gangue particle. The explanation of this phenomenon involves (1) the conception of potential energy at all contacts of dissimilar physical phases, and (2) the law of least potential energy.

The first conception is stated as follows: At the contact of any two unlike, distinct physical phases, there exists a certain definite amount of energy per unit of area. This conception has been proven for any number of contacts. Its effect is illustrated qualitatively by the floating needle previously mentioned.

The law of least potential energy states that every physical system tends toward a condition in which the potential energy of the system is least. Familiar examples of the manifestations of this law are the tendency of water at the top of a hill to flow to the bottom and reduce its capacity to do work; in other words, reduce its potential energy; and the tendency of a stretched elastic or spring to decrease its own length, in other words—reduce its ability to do work, or come to a condition of least potential energy.

Bearing these two conceptions in mind it will be seen that the spreading of oil on the surface of the sulphide, thereby displacing water from that surface, is a manifestation of the principle of least potential energy. Further, we may conclude that the energy of the oil-sulphide interface is less than that of the water-sulphide interface since such a condition only would allow for the spreading of the oil. Similar reasoning leads us to the conclusion that the energy of the water-gangue contact is less than that of the oil-gangue contact.

If, now, we consider an oil-water boundary as the fixed part of a system composed of oil, water, sulphide, and gangue and the sulphide and gangue particles the movable parts of the system, and if we consider the solid particles presented at this boundary, the sulphide will tend to pass into the oil because such a position will represent for it the position of least potential energy, while the gangue will tend to pass into the water for the same reason. If, then, we can form a system of oil, water, gangue, and sulphide, in which all the sulphide and gangue particles are presented at the oil-water boundary for sorting, we shall have a means of selecting the gangue from the sulphide. And is, after such selection has taken place, the selecting surfaces with their selected sulphide can be separated from the body of the pulp containing the worthless gangue, we shall have completed a process of concentration. This is done by introducing into the sulphide-gangue-oil-water system a large number of air bubbles. Whenever one of these bubbles, in its movement through the pulp, comes into contact with an oil globule, the oil will spread over the air-liquid interface under the action of the law of least potential energy, for the reason that the energy of an oil-air interface is less than that of an air-water interface, and an oil-air contact represents, therefore, a state of less potential energy than an air-water contact. At the surface of each such air bubble in the pulp there will be an oil-water interface for sorting sulphide from gangue and the sulphide will move into the oil away from the body of the pulp. As the bubble moves to the surface of the pulp through the influence of gravity the oil with its load of sulphide will move with it. At the surface the mass of sulphide-loaded bubbles will persist to form a froth.

Froth Formation.—The mechanics of froth formation can best be explained by the following series of experiments. Persistent bubbles, hence a froth, cannot be formed in water alone. This is due to the fact that a pure-water film cannot, except by the merest accident, come to a state of equilibrium between the force seeking to distend the bubble and the surface tension. Neither can a froth be formed with pure oil. A mixture of oil and water can be made to froth slightly on account of the fact that a film forms at the interface between the oil and water which has a sufficiently high viscosity to overcome the tension of the oil-air and oil-water interfaces. When air is introduced, into such an oil-water medium, into the water layer, the bubbles rise to the interface between the water and oil, and here immediately the viscosity of the interfacial film is apparent in the way in which the film is drawn up by the air bubble. Eventually, the bubble breaks free and rises through the oil layer with a coating of water. This bubble rises to the surface. Shortly, the water sheath surrounding the bubble falls away, and drops back through the water layer. At the same time

there exist, dispersed through the oil, tadpole-shaped water drops. The water drops are conclusive evidence of the high viscosity of the interfacial film between oil and water. It is apparent of course that, since the energy of the interface between the oil and the water is a constant, the drop will tend to assume the condition of least potential energy by reducing its surface to the minimum, and the sphere being the body of least surface for a given volume, the drop will tend to assume a spherical shape. But, since the drops do not assume the spherical shape under the influence of surface tension it must be further apparent that their tendency to do so is opposed by some other force. This force is the viscosity of the interfacial film. Such a viscous film aids a bubble with an oil-water film to persist and is one of the causes of the temporary persistence exhibited by bubbles formed from a mixture of oil and water. The other cause of such persistence is the variable surface tension of the oil-water film. Such a film when subjected to a stress greater than its surface tension can resist, stretches and in so doing increases its area. The concentration of the substance of lower surface tension (the oil) per unit of area now becomes less and the resultant tension of the film increases. If the increase is sufficient to balance the external stress, equilibrium is attained and the film persists.

Stabilizing Films in Froth Formation.—In the flotation process this variability in surface tension is but a small factor in the stability of the film. The great factor is increase in viscosity, due, however, not so much to the viscous films at the oil-water interfaces, as to the introduction into the bubble films of finely divided solid matter. The stabilizing effect of finely divided solid matter on liquid films can be illustrated in the laboratory by floating a match on the surface of a beaker of water; placing a chip at the side of the beaker, then causing the match to revolve slowly, when it will be seen that the match can be turned around the vertical axis of the beaker without any corresponding motion of the chip. If, now, the surface of the beaker is dusted over with finely divided solid matter, a film forms, and when the match is again caused to rotate, the film and the chip, in fact the whole surface of the contents of the beaker, move with the match. This, of course, can be due only to the increase in viscosity of the surface film. The surface of an oil-coated air bubble in a pulp, after it has picked up its load of solid, is similar to the water surface just described.

No physical principle other than the law of chance need be appealed to in order to explain the meeting of oil globules with air bubbles, or the meeting of sulphide particles with oiled bubbles. There are about three hundred million solid particles of the size that we are considering formed from every cubic inch of solid matter. In a flotation pulp (mixture of ore and water as sent to the machine) there are at least sixty million particles per cubic inch. The opalescence caused by agitating together the water and oil before addition of the ore will show

that the oil is as thoroughly dispersed. Given, then, at least sixty million particles of solid and sixty million oil particles per cubic inch, and introduce air bubbles in the same or greater number, and it is obviously unnecessary to appeal to any mysterious forces to bring about a meeting.

To summarize then; we bring to the machine a freely flowing mixture of ore and water, the pulp; we add oil; agitate to disperse the oil through the pulp; agitate further to introduce air in finely divided form; and agitate yet further to cause the air bubbles to come into contact with oil and sulphide particles. The oil spreads over the surface of the air bubbles because by so doing it lowers the potential energy of the oil-air-water system. A sulphide particle subsequently meeting the oiled air-bubble passes into the oil film because in this position it is in the state of least potential energy, and the gangue particle, meeting the oiled air-bubble, passes back again into the water because this position for it represents the state of least potential energy. The sulphide-coated, oiled air-bubble rises through the pulp, due to its smaller specific weight, and upon reaching the surface of the pulp persists, on account of the high viscosity of the solid-loaded film surrounding it. The froth formed by the persistent bubbles is scraped off and a separation is thus made between the sulphide and the gangue originally present in the ore.

CHAPTER IV

OILS AND THEIR USES

The oils used in the flotation process are many and their qualities vary within extremely wide limits, so that it appears that with ores of certain classes certain oils may be used, while with different ores oils having different qualities must be made use of in order to produce successful results. It has seemed apparently true that any particular kind of oil is not necessarily applicable to all ores, or perhaps it would be better to say that the same oil will not produce equally successful flotation results with all ores. That has been the theory heretofore and is an assumption that most metallurgists have taken for granted and have not seriously questioned. Apparently, however, the statement is not so broadly true as has been believed, and there are some who go so far as to say it is not true at all. The belief is not rare that an oil that will successfully promote flotation concentration on any particular ore can be used in making flotation concentrates on another ore, if the other conditions are so varied as to make them applicable.

Of course it is not intended to make the statement that all oils will produce satisfactory flotation froths, for it is well known that some oils produce good froths while others do not, but the point desired to be brought out is that an oil that makes a good froth in one case will produce froth in another case if the proper conditions are provided; that is, as to the nature of the pulp, solution, and combination of the two. The conditions of oil quantity and the presence or absence of a high degree of agitation must also be properly arranged to suit each case.

The Purpose of Oil in Flotation.—The exact function of oil in flotation is a point that has not yet been definitely settled. The primary function, which has been clear since the inception of the flotation process, is that the oil serves to reduce the surface tension of water and thus provide for bubbles and froth of more lasting character than could be formed without its use. In practice some operators have brought out the theory that only with

the lasting bubble in the froth can the best results in flotation be obtained. If that, however, is the only use of oil, what shall we say about those surface tension and gas-bubble processes in which no oil is used, such as the Macquisten process, the Wood process, the Potter-Delprat, and others of like nature? In these cases flotation is successfully accomplished without the use of any oil whatever. On the other hand, the flotation processes that are used on the largest scale and the ones that are achieving the greatest success are those in which oil is used. It seems evident, therefore, that in addition to the function of reducing surface tension, which the oil undoubtedly does accomplish, there must be further work that it performs. One of the properties of oils, it may be pointed out, is their preferential adhesion to some minerals rather than to gangue particles. This is a fact that has been emphasized by a large number of experimenters and one that has been accepted as true up to a comparatively recent time. Now there are many who believe that there is no selective action whatever and that the difference in action of the oiled minerals and gangue particles is due to other forces altogether. In spite of this suspicious attitude, however, it seems almost certain that there is a certain preferential action, the oil adhering to mineral particles, particularly of a certain kind, rather than to the siliceous gangue particles of an ore.

Preferential Oiling of Mineral Particles.—A careful examination of a pulp containing oil that has been added for flotation purposes will show that the majority of the mineral particles are more or less oiled, while the great majority of the gangue particles do not carry oil. This is subject to some qualification, however, for oils of certain classes—vegetable frothing oils, for example—are to be found in the gangue and not in the mineral concentrates. Such condition is probably due to the solubility of these oils in water, since they are not to be found directly in contact with gangue particles, but in the pulp tailings. This subject will be treated in further detail later. Whether this adherence is due to preferential action between oils and mineral, one cannot at the present time definitely say, but the fact remains nevertheless. As to why the oil tends to prefer the sulphide particles, we cannot be sure. The reason probably depends upon the adhesion, which is one of the few forces of nature upon which little light has been thrown so far by scientists. As a matter of fact, there are perhaps few who know

much about adhesion, while to the majority of even the scientists the reason for it is as unexplained as, for instance, gravitation. We can certainly say nothing certain about the oiling of mineral particles that will throw any light upon the preference, if such exists, for the oil to coat the mineral particles rather than the gangue particles.

It is a well-known fact that there are surface forces that may change completely the tendency of bodies to act in any particular manner. If we assume it to be a basic fact that mineral particles do become oiled in preference to siliceous gangue particles, it is not so difficult to imagine that there would be a tendency between the oiled mineral particles and the oiled gas bubbles to remain in contact with one another, the buoyant tendency of the gas then tending to lift the mineral in the froth. The well-known fact that solid particles carry electric charges, and that these charges in the mineral and gangue particles are or may be of different kinds, should be borne in mind in the elaboration of any theory of flotation or the function of oils in the process. The additional fact that these particles with electric charges migrate in the electric field, their direction varying according to the charge they carry, must also be remembered and may serve as a hint toward the preferential separation of minerals and gangue. In an ordinary dilute pulp this migration may be found to account for the fact that the minerals become more or less separated from the gangue, and may also help to explain why the minerals will take an oil film more easily and promptly, even if not to the entire exclusion of the gangue particles.

An objection to the electric basis for a theory of this kind, however, is found in the statement by authorities on colloids that this difference in electrostatic charge is a property of colloids only and does not apply to particles of practical size in suspension. Such particles, say these authorities, all carry charges of the same kind, and there would therefore be no tendency toward separation unless the oil film on mineral particles tends to change the character of the charge or to insulate the particle.

The surface films on particles will also have an important effect upon results. Surface films may be of four kinds—solid, liquid, gaseous, or electric. In solid surface films we have as an example the limonite coating formed on various mineral particles—for instance, bornite—which more or less effectually prevents

it from being successfully separated by flotation. It may be noted that if this surface film is changed or removed, flotation of the particles may take place. This is accomplished in practice by using an acid solution, or electrolyte, which removes the oxide, thus enabling the particles to be floated. The removal of this solid film may permit the oil film to be formed on the particles, which may account for its ready flotation under the changed conditions.

The Functions of the Oil Film.—Thus we see that there are at least two functions of the oil: First, to reduce the surface tension of the liquid so that a more permanent gas bubble may be formed, and, second, to oil preferentially the mineral particles so that some other effects, yet to be discussed, may be permitted to take place. The fact that the introduction of certain substances into the pulp solution may change entirely the effect of an oil is significant. Oils that produce a perfect froth under normal conditions may be so influenced that they will not produce a froth at all if certain substances are introduced into the solution. It may be noted that all of these substances are defloculating agents—that is, such substances as will permit the oil to become emulsified and maintain itself in that condition. While it may still retain the property of reducing the surface tension of the water through solubility or miscibility, and may permit the formation of bubbles (although this is not usually the case), the emulsified oil is still in such condition that it apparently cannot effectively oil the mineral particles, or perhaps any particle, so that its function will not be sufficiently well performed to enable it to be of use in promoting a flotation concentration, as has been pointed out by Van Arsdale, mention of which is made in the chapter dealing with the theory of flotation.

As to the service performed by the oil film on a mineral particle, reference to recent discussion on the theory of flotation indicates that a prime requisite for the flotation of a particle is the presence of a surface film having the property of refusing to be wet by water. The oil usually provides this film. The surface-tension theory of flotation necessitates the presence of this non-wet surface.

To provide a non-wet surface on the mineral particle, the oil used to supply the film must be such that it will not be wet by water, or some constituent must be added to the water that will

prevent it from wetting the oil. Most oils will not be wet by water, but some of them will, and the latter are not suitable for use in flotation unless something be added to the water to prevent the wetting action. Some acids supply this quality. If we attempt to explain the quality of an oil to be wet, or not, by water, we come again to the insurmountable barrier of adhesion phenomena, which alone can explain that quality.

It is to be remembered, however, that in spite of all this talk about oils and their effect, they are not the only materials that can form a non-wet film on solids. There are some solids that will do it; perhaps some gases, and not impossibly, electric surface charges. It has been shown that colloidal films of some kinds may be so manipulated as to produce this result.

Oils in general, then, may be divided into two groups, the first containing those oils whose duty it is to provide the non-wet film on the surface of the particle; and frothing oils, which serve to make the froth possible by reducing the surface tension by water.

Coal Tar as an Auxiliary in Flotation.—It is unfortunate, but true, that at the present time the realization of the functions that oils have to perform does not give us any advance information as to what kinds will be suitable to certain ores. So far it is altogether a matter of practice, the old rule of cut-and-fit being the only one from which much practical information can be derived. Every flotation plant finds an oil particularly suited to the ore being treated if the experiments are carried far enough. There seems to be no one oil that will give equally satisfactory results in every case. Coal tar may, however, become a universal oil when employed as a base for making good working mixtures of oils.¹ Until the latter part of 1914 the principal oils utilized for flotation in this country for copper ores were wood distillation products and cresol. The price of cresol, about 30c. per gal., has made its use costly, even though as little as two pounds of it per ton of ore is required, as compared with coal tar, which costs only about 5c. per gal. Later, other coal-tar products less expensive than cresol came into use, and finally coal tar itself aided by some frothing agent, such as pine oil. Pine oil, however, is also quite expensive, and subsequent experiments show that coal tar with from 10 to 24% creosote gave equally good results. Until these experiments had been performed, coal tar was not con-

¹"Use of Coal Tar in Flotation." William A. Mueller, *Eng. and Min. Journal*, Oct. 9, 1915.

sidered as a flotation agent in this country, nor was it ever experimented with here, although it had been mentioned in some English patents. The experiments referred to led to its adoption in one large mill, and many others are now experimenting with it and getting good results.

There is considerable difference in the behavior of different tars, and it might be said that while classed as coal tar, the usual run of this material is really gas tar, or tar resulting from the manufacture of gas from coal. These oils are byproducts and naturally can be bought cheaply. A number of experiments have been carried out using this oil as a basis on ore of a porphyritic nature carrying about 1.6% copper, mostly chalcocite, with some chalcopyrite and about 0.15% oxide. Coal tar does not froth very well, producing a lean froth, although one of good quality. Pine oil is employed in combination with the coal tar, the pine product being a good frother and the combination of the two producing a carrying froth that gives good results in concentration. A mixture of 80% coal tar and 20% pine oil produces first-class concentrates. The price of pine oil, however, is high, and its use should be avoided whenever possible. The coal-tar products—cresol, creosote, etc.—which formerly served alone, are now mixed with the coal tar in proportions up to about 20%. The mixture thus obtained is excellent in both the pneumatic and agitation types of machines.

Use of Coal Tar with Frothing Oil.—When coal tar is used long enough, it alone gives low-grade tailings together with high-grade concentrates. It requires, however, that the ore be held in agitation such an inordinately long time that the practice is not considered satisfactory, and it is usually better to add some frothing oil along with it to produce the results in a shorter time. Some tars may not give any results at all when used alone, but as soon as a very little of such an oil as pine oil is added, good results are often obtained immediately. The amount of oil will vary according to the ore, grade of concentrates, and tailings desired; 0.005% of it will work a marvelous change in its effect upon some ores.

As the amount of tar increases in proportion to pine oil, the amount of concentrates increases and the mineral content of the tailings decreases. This may be on account of the proportion that coal tar bears to the mixture, or it may be because of the amount of oil resulting from increasing the amount of coal tar.

Rough experiments seem to indicate that it depends upon the amount of mixture employed.

In a description of flotation tests on ores from Bisbee and Cobalt, Herbert J. French makes some interesting observations on oils.¹ In experiments on Bisbee ores the character of the material is described as porphyritic, the copper existing almost wholly as cupriferous pyrite, chalcocite, and malachite. The total copper amounted to 6.04%. After testing the ore in the slide machine, the tailings were analyzed for sulphur, whence the percentage of extraction of the sulphides could be computed.

Tests on Various Kinds of Oils.—Several oils, such as fish, sperm, whale, corn, cylinder, pine, pine-tar, lard, and cotton-seed oils, were tried under fixed conditions. Both qualitative tests, by shaking oil, acid, ore, and water in test tubes, and quantitative tests on the slide machine were made. Of those oils available it was found that pine-tar oil gave the maximum extraction with the minimum weight of froth. Pine oil and fish oil both gave good results, but the froth, in addition to being of non-selective character, did not seem so lasting with either of these as that given by pine-tar oil. The character of the froth obtained by the other oils mentioned did not seem to indicate successful operation; hence the pine-tar oil was selected as the most desirable. In the experiment carried out at room-temperature on 100 grams of ore, the pulp containing sulphuric acid, with a 3-min. period of agitation, it was found that the extraction increased with the weight of oil added. A certain point, however, is reached beyond which more oil does not increase the extraction sufficiently to warrant its use in larger amounts. It is also evident that with the increase of oil, other conditions being the same, more of the gangue floats. In other words, the action of the oil becomes non-selective with increase above a certain point.

In another experiment, varying the acid, it was again found that there is a point beyond which further addition of acid is useless. When time is taken as a factor for variation, it is found that the longer agitation is continued the better the results. And again, when the temperature is the factor to be varied, it is found that from 50° to 65°C. gave as good extraction as could be had.

Experiments were then made upon ores from the Coniagas

¹ *Columbia School of Mines Quarterly*, November, 1914.

Mining Co., Cobalt, Ont. This ore contains about 13 oz. of silver per ton, mainly in the form of pyrargyrite, proustite, argentite, dyscrasite, and native silver. The procedure was about the same as for the Bisbee ore, but in this case the tailings were analyzed for silver. In the first test, which was made with different kinds of oil, pine-tar, fish, sperm, corn, whale, cylinder, rape-seed, lard, and cotton-seed oils were tried, and it was clearly demonstrated that pine-tar oil was far superior to the others. Rape-seed oil gave a comparatively good extraction, but it was noticed that the froth was not so lasting as that produced by the pine-tar oil. The amount of oil was next varied, and in this case there was a recurrence of the phenomena of critical quantity. It was shown that beyond $2\frac{1}{2}$ grams per 100 grams of ore, no benefits were derived from additional oil.

Examples of the Use of Mixed Oils.—At the Gold King mine,¹ at Gladstone, San Juan County, Colo., a low grade silver-lead ore is treated by flotation, the oils used being carbolic and creosote. Carbolic oil is fed to the pump used to hoist the pulp to the stock tank, while creosote oil is fed to the first compartment of the agitator and pine oil is fed to the fourth compartment of the agitator, 300 c.c. of total oil per ton of pulp being used. In this case no acid is found necessary.

At a flotation mill in Mexico,² the best flotation agents among those tested were pine oils, a low-grade giving as good results as the better varieties. For the best work in flotation it was necessary to have pine oil present to the extent of 0.6 lb. per ton of ore. In actual plant practice, where the water is returned again and again to the top of the mill, the consumption of oil would reach about $\frac{1}{4}$ lb. per ton of ore. Pine-tar oil is much cheaper. It gives good extraction, but the grade of concentrate is low. Cresylic acid, when used with pine oils, increases the extraction about $\frac{1}{4}$ oz. of silver per ton. This hardly pays for its use. It may be said in connection with cresylic acid, however, that aside from any function it may have in assisting the actual formation of froth, it has the property of reducing the adhesion between oils that would ordinarily be wetted by water, and the water itself. Therefore the function of cresylic acid in the flotation process would seem to be in its quality of robbing water of the ability to wet certain oils.

¹ *Eng. and Min. Journal*, Oct. 16, 1915.

² "Flotation in a Mexican Mill," *Min. and Sci. Press*, July 24, 1915.

It may be noted that so far in the review of the oils used coal tars have proved themselves to be poor frothers, and to get a sufficient volume of froth to make good recovery it is necessary to add something else as a frothing agent. Pine-tar oil, pine oil, creosote, etc., are the ones usually required. At the Inspiration mill, in Arizona, the flotation mixture is 80% crude coal tar, 15% light oil or creosote, and 5% pine oil. At the Daly-Judge, a mixture containing 40% crude coal tar, 40% creosote, and 20% pine oil is used. In the Coeur d'Alene country, wood creosote is used on zinc ores, and on some copper ores plain turpentine will work, but pine oil is better. The latter seems to have the property of making a brittle froth that does not seem to last long on reaching the surface of the floating machine, an advantage in many cases. Creosote and light oil makes an elastic envelope that will expand to great size before bursting. The pine-oil bubbles will rarely be over $\frac{1}{4}$ to $\frac{1}{2}$ in. in diameter. Castor oil, olive oil, oleic acid, palm oil, sperm oil, and other oils of a lubricating nature have in general been replaced by oils more or less soluble or miscible in water, such as turpentine, pine oil, eucalyptus oil, and all other coal- and wood-tar distillations. Extremely volatile oils, such as naphtha, gasoline, ether, and alcohol, do not, apparently, have any effect themselves aside from the property they have of dissolving oils and thus making them more soluble or miscible in water.

On some of the mixed lead-zinc ores of the Coeur d'Alene country a mixture of three parts pine oil and one part creosote seems to work very well. The quantity used is about 0.5 to 0.7 lb. per ton. At the Silver Peak mill, Yerranderie, N. S. W., Australia, the best flotation results were obtained by the use of two oils. *Eucalyptus amygdalina* was used as a frothing agent and a dark lubricating-oil mixture as a carrier.¹ The pulp was heated to 120°F., at which best results were secured. The consumption of eucalyptus oil was $\frac{3}{4}$ to $\frac{1}{2}$ lb. per ton, and of the mineral-oil mixture 2 to $2\frac{1}{2}$ lb. per ton.

The Qualities of Pine Oils.—In discussing flotation oils up to this point it will be noticed that there have been references to products from pine distillation, sometimes called pine oils and sometimes referred to as pine-tar oils. It might be assumed at first hand that this was simply two different ways of speaking of the same thing, but such is not the case, because the products

¹ H. Hardy Smith, *Eng. and Min. Journal*, Dec. 11, 1915.

are different. Pine oil is produced by the steam treatment of pine wood, cut up into small pieces, while pine-tar oil is obtained through the destructive dry-heat treatment of the wood. According to the specifications of the United States Navy Department, pine oil is defined as "a properly prepared light, straw-colored oil, produced by redistillation of heavy, high-boiling-point fractions, resulting from the steam distillation of wood turpentine, and having a strong aromatic odor resembling turpentine." This oil should have a specific gravity of not less than 0.930 or more than 0.945 at 60°F., and upon distillation should show at least 75 % distilling between 200° and 220°C. The same authority, defining pine-tar oil, says: "A properly prepared distillate from straight pine tar, which latter has been obtained by destructive distillation of pine wood." It will be clear, then, that the two products are different and are not necessarily interchangeable in use where experiments have been made upon only one.

Only a small amount of pine oil is produced, not nearly sufficient to supply mining demands. High-grade pine oil is also used for medicinal purposes and, pure, commands from 60 to 80c. per gallon.

Long-leaf pine produces a number of oils, just as petroleum gives a variety of products. The fact that pine oil, which has a short range of distillation and is heavier than turpentine, was on the market and was used in flotation mixtures has led to the belief that no other oil coming from the pine tree would serve the purpose as well. Experiments, however, prove that oils heavier than pine oil, when properly mixed with other oils, produce a better froth than pine oil. By the usual methods the distillation of pine oil is not carried beyond a specific gravity of 0.940. Stills are not equipped to carry it higher. Medicinal practice demands an oil distilling between 190° and 220°C. Experiments show that the oils distilling above 220°C. are the best for flotation.

Oils are now produced of a specific gravity beyond 0.970, but they are too thick to handle easily, and are mixed with lighter oils, resulting in two grades, 0.875 and 0.900 specific gravity. They produce a splendid froth.

Some tests have been made on pine oils, in one of which the oil was noted as orange in color and having a specific gravity of 0.945. In this experiment 2 lb. of the oil was used per ton, and

2 lb. of acid was added to the pulp and agitated one minute before the oil was added. It was found that by using the oil thus, a good product was obtained, but by making two or more operations on the same pulp, adding 1 lb. of the pine oil first and then at the second operation a pound of oleic acid, a high-grade product was also produced. In testing two or three kinds of pine oil in combination with oleic and other oils, it was found that a consistently good product was obtained each time, and that the percentage of extraction increased with the number of operations that were performed on the pulp, up to the point where practically all the sulphide had been removed from the oil.

In describing the flotation process at the Oneida Stag mill, at Idaho Springs, Colo., R. W. Smith¹ points out that oil at the rate of $2\frac{1}{2}$ lb. per ton of dry feed was added. The oil cost was materially reduced by mixing three different kinds—in this case consisting of one part of mixed crude wood tar, fairly cheap; one part of spirits of turpentine, more expensive; and four parts of a cheap, local mineral oil. The mineral oils constitute a cheap base for carrying the other oils. It is not always possible to find a mixture that will give a clean, high recovery and at the same time make clean products. By mixing in this way, however, the cost per gallon for oil on the ground was reduced from 45c. to 23c. This reduces the cost of oil per ton of crude mill feed from 6c. to 3c.

Oils Used at the Anaconda Mill.—In the description of flotation at Anaconda, the kind of oil used for their flotation has been noted by Laist and Wiggin.² A product that has heretofore been little used in flotation, but has been the subject of some experimentation, is a kerosene sludge-acid that has been employed by Minerals Separation and is called M. S. 37. It was used practically all through the experimental work at Anaconda and gave highly satisfactory results. Although exposed to the weather in iron drums during the winter months, it gave no trouble by becoming too viscous. One barrel of oil from the Western Oil Co. was tested during February, but did not give as satisfactory results as the oils from Standard Oil Co. and the Union Oil Co. of San Francisco. One barrel of oil from the Midwest Refining Co. was tested during February, but did not give the results obtained on the other two. From the Producers Refining Co. came

¹ "Flotation Replaces Cyanide," *Eng. and Min. Journal*, Jan. 15, 1916.

² *Bulletin A. I. M. E.*, March, 1916.

an oil that could not be tested, as it had solidified in the barrels and did not become liquid after standing for days at a temperature of 60°F. or better.

During the experimental work creosote oils were tested with varying results. The creosote from the Cleveland Cliffs Iron Co., of Marquette, Mich., is obtained as a byproduct in the manufacture of charcoal from hard pine. This is known as M.S. No. 33, the number given to it by the Minerals Separation Co. It was used practically throughout the entire work with highly satisfactory results. Creosote from Pensacola Tar and Turpention Co., of Gull Point, Fla., was used in considerable quantity in experimental work. It gave fully as good results as the Cleveland Cliffs creosote. It is made from Georgia pine and is known as No. 400 by the Pensacola Company. A barrel of creosote from the Crichton Pine Products Co., of Alabama, was tested and gave good results. A test was made using a barrel of tar creosote from the Butte Gas Works, and this gave fairly good results. The test was too short to be conclusive, but it was thought that the tar creosote would give satisfactory results.

In testing the Callow machines a mixture of oil containing tar creosote from the Barrett Manufacturing Co. was used with good results. In the laboratory experiments it was found that tar creosote gives good results. Creosote from the timber-treating plants at Rocker comes from the J. F. Lewis Co., of Moline, Ill. It did not give good results, although the test was too short to be absolutely conclusive.

The Acid-Hydrocarbon Products.—The reference to the kerosene sludge acid introduces a new oil, or hydrocarbon product, that has not been spoken of before, the manufacture and production of which is not well known by flotation men. The product is one from the distillation of petroleum. The burning-oil stocks from the steam stills, consisting of from 65 to 75% of the crude oil charged, are treated at the rate of 5 to 10 lb. of 66° commercial sulphuric acid per bbl., in order to improve the color and odor and also to remove decomposition products which cause the flame to smoke when burned in lamps. The process is performed in agitators, using an air-blast for agitation. In order to get the full benefit of the acid, any water present must first be drawn off and a small amount of acid added in order to remove all the remaining water. This is agitated for from 20 to 40 min., allowed to settle from the oil and then drawn off. The remain-

ing acid is then added and the mass agitated for from one-half to one hour, allowed to settle from 3 to 5 hr., and drawn off. The acid treatment removes the tarry matter formed during distillation, also a large percentage of the unsaturated hydrocarbons and sulphur compound. The acid is turned almost black after treatment, and is known as sludge acid. It is allowed to settle from the oil and is then drawn off into tanks and delivered to the acid-separating plant for the recovery of the acid. This sludge acid is the product that has been found useful in flotation. It will be observed that it comes from the treatment of light burning oils. A similar process is practised upon the lubricating oils, but the sludge acid from these heavier oils is black, asphaltic in nature, and very viscous—almost a solid mass. It, with the sludge from the burning oils, is sometimes agitated with water in a lead-lined separator, the weak acid recovered is concentrated in lead pans, and finally in stills of cast iron to 66° Baumé, and then used over again. The residue from this treatment is usually used for fuel or worked up into pitch. It will be seen that the use of these sludge-acid oils may possibly combine the features of having the oils necessary for reducing surface tension together with the acid required in the treatment of some particular ores.

In a test made on some oleic and coal-tar oils, a cycle of four different operations was carried out on each. The test was performed at a dilution of 4:1 and at a temperature of 45°C., the ore being a typical blende combination with heads averaging 15½% Zn. The additions consisted of the oleic oil and 2 lb. of acid per ton. Clean concentrates of 45.6% zinc were produced, with clean tailings containing 1.3% zinc. In using the straight coal-tar oil the same feed produced clean concentrates of 49.5% and clean tailings of 1.2% zinc. In each case there were four operations performed.

Pine Oils and Petroleum Products.—There are several different kinds of pine oils, made by different manufacturers, and although the qualities are in general approximately the same, some difference may be found in particular products. Taking for example Spiritine and the Atlantic as two different examples of pine oils, it has been found that they both give good froth. It may be noted that when pine-tar oil is used, a long contact with an acid solution will be likely to cause the froth to break up and the concentrates to drop out of it. For this reason, where long contact is essential it is believed, at least in some cases,

better to take an oil in which the pine-tar product does not exist. One of the two pine oils has been found to contain pine tar, and although it gives an excellent froth on short agitation, long contact with acid will, as has been said, cause the concentrates to drop out of the froth. Where acid is not used, however, pine-tar oils are particularly advantageous and make splendid froths. A large number of tests have been made with acid petroleum products, some of which make good flotation oils. In tests on Texas crude oils, most of them have been found to give good results in combination with a frothing oil. The frothing oils used in combination in this case were oleic, stearic, pine, and pine-tar oils. The Texas crudes are usually known as A, B, C, D, E, and F. Combinations containing half of these Texas oils and half of one of those that have been mentioned as frothing agents were tried, and most of them gave very good results. The Texas crude "A," in proportion of one-half to one-half of stearic, however, requires considerable agitation. This agitation is more than is required when the other combinations of oils are used. The combination of the Texas crude "C" with oleic acid, in the proportion of half and half, froths pretty well, but the froth did not show much strength nor stand up well.

The Minerals Separation oils have already been referred to, but in some cases they are used in combination with pine and pine-tar oils, as well as with oleic and stearic acids. In a few cases, however, tests of a mixture with oleic and stearic acids did not give good results, but there was probably some local reason for such an outcome.

Thorough tests have also been made of such products as crude oil, fish oil, and many others. Fish oil gives good results, and a combination of fish oil and oleic acid does too, although it shows some tendency to carry up siliceous matter with the mineral. Fish oil and pine oil in combination do not seem to do any good at all. In the experiments made on this combination, no results were secured. Stearic and oleic acids give good results, as also does stearic acid alone. Beaumont oil with tar oil and with oleic gives fairly good results, and some paint oils may also be successfully used. Coal-tar oils have already been mentioned, as has also their combination with pine oil and other frothing agents.

Tests on More Unusual Oils.—Pine oils and pine-tar oils have been tried in combination with cotton-seed, lard, nitrobenzol, turpentine, oleic acid, stearic acid, corn oil and other hydro-

carbons. The results seem to vary with the kind of ores treated. In many cases a good result can be obtained with blende ores, while the same oil combination will not work at all on copper. At the same time, if the oil mixture is changed and also its proportion to the dry ore itself, the same oils, alone and in combination, may be made to do good work on the copper ores. The same applies to the treatment of any other kind of ore where changes in oils seem to give different results. In spite of the efforts that have been made to discover some rule for the application of various oils to different kinds of ores and in spite of the careful study given to the matter, no general principles can be announced. At many times the experimenters have seemed to be on the track of a logical explanation, but with the formulation of rules, numerous variations and objections to it have cropped up, so that at the present time there is really nothing that can be laid down as specific.

Tests have been made on corn oils, and the corn oil in combination with pine, pine-tar, oleic and stearic acids has been tried. All of them give good results under some circumstances and poor results under others. It might be said in general that any oil will form a good flotation product if the proper combination with its complementary oil is used. By this is meant that if a carrying oil is used, it must be supplemented by a small quantity of a frothing oil. The frothing oils and the carrying oils are complementary to each other, one supplying the qualities that the other does not possess. There are some oils that combine the qualities of carrying and frothing agents, but as a rule they are so expensive that they cannot be used alone, and it is found cheaper to use a large quantity, comparatively speaking, of the cheaper carrying oil and a much smaller quantity of the more expensive frothing agents. Of the Texas crude oils there seems to be none so far tested that can be depended upon to give satisfactory results alone. They all have an asphalt base and bring forth a light, rich, unstable froth. In combination with either pine, oleic, or pine-tar oils they give good results. They are graded according to their specific gravity, the lightest oil being the Texas "A" and the heaviest the Texas "F." The latter, in combination with oleic and pine in a proportion of one-third each, gave remarkable results on some careful experiments.

Comparison of many different kinds of pine oils show that there is very little difference in the product of different manu-

facturers, most of them having about the same properties. Oleic oils have been tested in combination with corn, pine-tar, turpentine, and other oils, and also straight. In proper quantities they seem to give good results in every case. Oleic oils of different manufacturers have been also compared, and there seems to be very little choice between them.

Sagebrush Oil.—The use of oil extracted from sagebrush was suggested in papers read at the Arizona meeting of the American Institute of Mining Engineers, September, 1916, by G. H. Clevenger and Maxwell Adams. In both these papers the oil extracted from the variety of sage known as *Artemesia tridentate* is said to have power as a flotation agent.

The method of extracting the oil is very simple. The leaves, twigs and small branches are placed in an air-tight drum, having a capacity of about 27 cu. ft. Steam is admitted through a number of small openings at the bottom of the retort, and the pressure maintained at 20 to 25 lb. per sq. in. for three hours. The escape of the steam from the retort is regulated by allowing it to pass through a stop-cock into a condenser. The water in the receiver is drawn off from time to time and the oil, which is insoluble and floats upon the water, is thus collected. At the end of two hours most of the oil has been driven out, though traces continue to come over for a much longer time. By raising the pressure, the time required could probably be shortened and the yield increased, but the lack of laboratory equipment has prevented the carrying out of this experiment.

The stock wood, bark and branches contain no oil, the distribution of the oil being limited to the leaves and young shoots. There is a seasonal variation in the amount of oil contained. Samples collected on different dates gave the following amount of oil: May 1, 0.42%; May 27, 0.6%; June 30, 0.72%; Aug. 1, 0.9%; Sept. 10, 1.0%. The increase appears fairly constant from early spring, when the leaves first appear, until light frosts occur in the autumn. When the plant is air-dried there is some loss of oil, as the following data will show: Two 100-lb. samples were collected at the same time. One was distilled when green; the other was air dried for ten days before distillation. The green sample yielded 275 grains, and the dried sample 248 grams of oil, showing a loss of about 10%.

Miscellaneous Oils Tested.—Tests on a lot of miscellaneous oils have given, in general, good results, although some will not

work when economically possible quantities are used. Wood creosote works well and produces a good, thick froth, but has a tendency to carry silica into the froth. Spike oil was tried, but it usually does not seem to produce a froth that is tough enough for practical satisfaction. Rape oil produces a pretty good froth, fairly clean, although on continuous subsequent operations there appears to be a peculiar scum that interferes with satisfactory flotation. Nitrobenzol is too light for flotation, and no results can be had with it. Castor oil alone does not usually produce good results. In combination with turpentine it will work satisfactorily, but such large quantities of it have to be used (up to 8 or 10 lb. per ton) that it is not economically practicable. Cotton-seed oil and coal oil (kerosene) alone did not produce any practical results. Coal oil and pine-tar produce pretty good results, but require a large quantity of the oil in proportion to the tonnage treated. Turpentine alone does not produce satisfactory results, according to experiments, nor does the variation called turpentine oil. All the turpentines work fairly well, however, when combined with a heavy oil. The use of fish oil in large quantities gives fairly good results, but its extremely disagreeable odor would be a drawback. The trouble experienced when stearic acid is used alone seems to be that stearic brings the congealing point down, and this oil also takes more heat to render it liquid enough for use. It seems probable that coal-tar oils and coal tars themselves will probably be the most generally useful oils in flotation processes, being used as carrying oils and forming the bulk of the oils added. They must, however, be combined with some oil having frothing qualities, such as pine, pine-tar, or many of the other lighter oils already mentioned.

The practical result of a large number of oils tested seems to be that the only way to determine whether an oil is applicable for flotation of a certain kind of ore is to try it on that ore. And, it will be noted, it is advisable to try an increasing number of oil combinations and quantities. Since, as has been mentioned, it is totally impossible to outline any rule for forecasting the results to be given by oils of certain classes on ores of particular kinds, a large number of experiments is absolutely necessary to accurately determine such oil or combination of oils that can be used, and the minimum quantity that can be used to secure the greatest economical quantity of flotation concentrates.

Testing Flotation Oils.—Oils may be tested for their use in flotation plants and, in fact, no oil should be used until a thorough test has been made and its value for use in the plant proved. The great difficulty is that there is no standard way of making tests that will indicate the usefulness of an oil in flotation, so that the usual method is to choose a standard ore on which tests of all kinds are run and recorded, the effect of the oils being referred to this particular ore as a standard. This method has been brought out by J. Coutts.¹ He points out that the first thing to do with an oil is to measure its density for future calculation, as it will be measured by volume in the laboratory and there must be some method of transferring these volume units into weights.

Coutts classifies three different kinds of standard tests: First, one for mixed sulphides; second, one for differential separation; and third, one for the flotation of copper and iron sulphides. He believes that oils high in phellandrene are best for differential separation of zinc-lead-sulphide ores. Phellandrene is an oily constituent that leaves a gummy residue on evaporation. It may be tested for by evaporating oils at moderate temperature and examining the residue, or it may be detected by the polariscope. An oil giving a deflection by the polariscope of 60 or over is considered to have a sufficient quantity of phellandrene for use in copper flotation.

For testing the number of drops of oil in each cubic centimeter a small burette is recommended, the discharge orifice of which has been made smaller by heating and dumping it up. About 80 or 90 drops per cubic centimeter should be given at normal temperatures. The number of pounds of oil per ton of ore can be found, when the specific gravity and number of drops in one cubic centimeter are known, from the accompanying formula:

$$\frac{2,240 \times \text{sp. gr.}}{453.6 \times \text{drops per c.c.}} = \text{lb. of oil per ton}$$

The great difficulty in testing oils for flotation is that it is not yet known what quality it is that makes oils of value for flotation. The only way to find out if an oil will do for this purpose is to try it on the ore to be tested. Some laboratories have under way series of tests looking toward the determination of the

¹ "Aust. Min. Standard," Apr. 8, 1913. Reprinted in *Eng. and Min. Journal*, June 19, 1915.

flotation quality of oils, and also the matter of changing the quality so that they may be used for flotation successfully. In this case reference is not had to the mere mixing of oils to secure good flotation froth, but the actual changing of the quality of the oil itself by such means as dissolving different materials in it, sulphonating it, or modifying it by heat or other treatment. Some progress is being made on the production of oils of good quality from cheap oils that can be obtained readily in particular localities.

The Classification of Oils.—The New York Produce Exchange defines crude petroleum as follows: "Crude petroleum shall be understood to be a pure, natural oil, neither steamed nor treated, free from water, sediment or other adulteration, of the gravity of 43° to 48° Bé."

Since the production of the lower Baumé gravity oils, the limitations in regard to gravity have been relaxed. In order to ascertain whether or not the petroleum conforms to the foregoing rule, it is usually distilled in 10% fractions, and the Baumé gravity determined on each fraction. It is obvious that from the complexity of crude petroleum each succeeding fraction would show a somewhat regular graduation in specific gravity, whereas in cases of adulteration it would be more or less irregular. There are a number of tests that can be performed on oil to determine its constituents in oils of different gravities and also to determine the quantity of sulphur contained, the specific gravity of the oil, the water contained, and other tests. Burning oils are usually tested to show their flashing point. Lubricating oils are also tested in the same way to show their flashpoint, and in addition, their viscosity and their cold test—that is, the temperature at which they become solid. All of these are set forth in detail in the textbooks on the subject of oils and are too complicated to be treated here, particularly in view of the fact that they have no relation whatever to flotation so far as we know at the present time.

Aside from the petroleum there are many liquid fats and fatty oils in addition to the solid fats. In the first class there are the vegetable oils, which may be divided into drying, semi-drying and non-drying. In the second class are the animal oils, which are subdivided into marine animal oils, such as fish oils, liver oils, and blubber oils; and the terrestrial animal oils.

The solid fats are divided into vegetable fats and animal fats, and the latter are subdivided into drying fats and non-drying fats. In these classes come a host of different kinds of oils, many of which have already been mentioned in this article and hundreds of which have not. It seems that almost every vegetable or plant will produce an oil having some different qualities from every other. There are also animal oils and fats of different qualities and having different effects. Only a few of the large number are of interest in flotation, because only those few occur in such abundance as to be commercially cheap enough to be used in a commercial process of this kind.

Froths and Their Character.—The kinds of froths formed by the different oils used in flotation have been described by W. A. Mueller,¹ who has also considered the effects of different machines and conditions on froths. He points out that for studying the various froth characteristics of an oil a small testing machine works well and is handy to use. The froth produced by various oils differs with the machine in which it is made. In the Minerals Separation type, where the froth rides in the spitzkasten for some time before leaving it, there is plenty of time to collect, and the froth always lies in a "dead" state, only coming off as it is forced out by new froth from the rear. In this case the froth loses its characteristics so far as the bubble itself is concerned. In the Callow type of machine the froth is always in rapid motion and no dead froth accumulates. It is kept overflowing by the constantly rising bubble column formed. In the Cole-Bergmann and Flynn-Towne types of machines the froth acts in the same way. In these two, however, a deeper froth column is carried and this makes the bubbles slightly smaller. In the Hebbard type of Minerals Separation machine the froth is in more rapid motion than in the regular machine, owing to there being no settling compartment and also because of the introduction of air. There is no accumulation of dead froth except at times around the shaft of the agitators.

In adding oils for flotation it is of the utmost importance that they be well mixed with the pulp before flotation is attempted. If not thoroughly mixed, the oils kill froth rather than produce any. When a machine is in good working order and some raw oil suddenly gets into the pulp, the froth immediately seems to

¹"Froths Formed by Flotation Oils," *Eng. and Min. Journal*, July 1, 1916.

dissolve, leaving the pulp showing through the froth. In these cases the froth rushes back and forth in the machine, the bubbles become very brittle and finally disappear.

The practice of using a shallow froth is considered good by some metallurgists, but does not really aid except where there may be a temporary air shortage in the air type of machine. The usual depth of froth in the Callow type of machine is about 15 in.; in the Cole-Bergmann and Flynn-Towne machines it is from 24 to 36 in., while in the Minerals Separation type the froth level varies from 6 to 12 in. in the spitzkasten, and in the Hebbard type it runs from 6 to 10 in. The raw-oil condition in a machine can readily be distinguished from the high level of pulp condition by the depth of pulp from the top of the machine at the time the condition is noticed. The raw-oil action is detrimental to flotation and should be avoided. Cup greases and lubricating oils give the most trouble of this kind in plants.

The froth characteristics described apply to live froths such as are formed in the air types of machines. The agitator type, with the exception of the agitator-and-air type combined, usually has a spitzkasten or settling compartment in which the froth congeals and so loses its bubble characteristics. In these machines the froth characteristics are not so noticeable.

Characteristics of Pine-Oil Froth.—There are several grades of pine oil, but the various kinds have practically the same effect on the froth. Pine oil is rather soluble, and it is this characteristic that is largely in its favor, as its presence, once used, is always detected in the mill water as long as the tailings and concentrates are dewatered and the water reused. After using pine oil a short time, only a little of it need be added—just enough to make up for any small amount that may have been lost or carried away by the concentrates after dewatering.

The pine-oil froth is brittle and the bubble produced is small, the largest one being only about $\frac{1}{4}$ in. in diameter. A small amount of oil produces a large quantity of fine-grained froth. The bubble film is very delicate and, on rising to the surface, breaks immediately. Its burden of mineral, if there is any, is deposited on the bubble immediately following, and thus the mineral is carried along on the surface until it flows over a lip or otherwise is removed. Pine oil as a general rule is a rather poor mineral oiler, but a good frothing agent. For this reason it is used principally as a frother, although there are some cases

where it will take the place of a mineral oiler as well as the frothing agent. Its action is delicate, and at plants where pine oil is largely used, the machines will require much closer attention. It is a good mineral collector for ores carrying chalcocite and on some zinc ores. A pine-oil froth, in addition to being brittle, is also watery and at times, when there is too much of it present, it tends to make a dirty froth and low-grade concentrate. When present in these large amounts, the concentrate froth will often carry large particles of silica and gangue material. The froth, however, does not toughen up because of its excess oiling, and in this way pine oil differs from most others. The froth will be dirty, as is the case with other oils, and cannot be cleaned, but the latter is due largely to excess froth rather than to the toughness of the froth. By the addition of acid, or sometimes a caustic, its action as a mineral collector can often be increased, but it still retains the same characteristics of brittleness and delicacy.

Next to the pine oil the most important flotation agent is perhaps coal tar. The action of this has already been described, and it has been shown to be a poor frothing agent but a good oiler. It is seldom used alone. When it appears as a foreign oil in the machine it lacks the strong froth-killing power of the other oils, but lies on the froth in flakes, sometimes deadening the froth in the immediate vicinity and at other times seeming to mat the small bubbles together, forming a sort of hole in the froth toward which the bubbles in the vicinity have a strong tendency to rush. Arriving, they seem to be swallowed up much as a cork will act when thrown into a whirlpool. Raw coal tar in the machine is not so detrimental as the other oils, but it should be avoided. When in great excess coal tar has a strong tendency to separate, as has been noted, even when apparently well mixed. Even though it has no strong frothing power, when too much is used it will cause some excess frothing and a dirty froth, though not so easily as other oils.

The Action of Coal-Tar Creosote.—Coal-tar creosote is used mostly as a dilutant for the cheaper coal-tar oil and for this purpose it is very good. The bubble has a tendency to be rather large, ranging in size from about an inch in diameter to as much as 12 in. and sometimes even more. It is very tough and in breaking, the bubbles tend to expand rather than be destroyed altogether. With two bubbles side by side, the breaking seems to

take place in the film uniting them, so that where there were two bubbles at first, there is only one after bursting. This enlarging of the bubble, of course, has its limit, and we seldom see them much over 12 in. in diameter unless there is a great excess of creosote, when they may get to be of such size as to reach across the machine. On bursting, the mineral burden is delivered to the film of the neighboring bubbles with which they were in contact before bursting. The bubble has good carrying capacity for mineral. When too much oil is present, however, there is the same tendency to carry gangue, and this is usually at the expense of the mineral. The oil mixes rather easily, the action in the Pachuca tank being sufficient, although mixing in the grinding mills is at all times to be recommended.

Wood-Creosote Bubbles.—Wood creosote has some of the same characteristics of the coal-tar creosote, but differs widely in others. The toughness of its bubble is even more marked. The size of the bubble is different and the two oils are not interchangeable—that is, one cannot be used as a substitute for the other, as coal-tar creosote seems to work better on copper-bearing ores than the wood creosote and the latter seems to work better on zinc ores. Wood creosote is often used as dilutant for coal tar in the flotation of zinc ores. The size of the bubbles varies greatly. The characteristic froth shows small ones resembling the pine-oil bubble, together with the larger ones. There will be a large one about 3 in. diameter as a maximum; then there will be a group of smaller ones, ranging in size up to maybe $\frac{1}{2}$ in. diameter, which separate the larger ones from each other. The froth, like that of coal-tar creosote, is tough, and a small excess of oil makes a dirty and voluminous froth that is hard to break up. The bubble varies from this type when an excess of oil is used, tending to become larger and more elastic.

Turpentine is not used so much at the present time as when flotation first appeared in this country. The turpentine froth is persistent and elastic. It has a tendency to hold everything that comes in contact with it, and for this reason it was formerly used to bring up the middlings and was added after the concentrates were taken off. For this purpose it was good, but the froth is so tough that it is hard to break down, so that cleaning it is difficult. Usually the middling produced by means of the turpentine could not be cleaned to any appreciable extent. The turpentine bubble ranges in size from $\frac{1}{2}$ in. to 2 or 3 in. diameter.

It is elastic, and the volume of froth produced is always excessive and, as a rule, low-grade. For this reason it is not used much at present.

Cresol, or cresylic acid, was formerly the chief frothing agent used in the flotation of copper minerals, but its use is now limited to small proportions in the mixtures with coal tar. The cresol bubble is of medium size—that is, from $\frac{1}{2}$ to 2 and 3 in. diameter—and as a rule the bubble is not too brittle and not too tough, but made a good froth to handle. The nature of the oil, however, made it disagreeable to handle and also somewhat dangerous; this, together with the bringing out of coal tar and the advance in price of cresol, gradually brought it to a point where only a comparatively small amount is now used.

The oil was a good frothing agent as well as a good mineral collector, and its use was formerly almost universal. The froth is rather easy to break down, and the concentrate produced is easily cleaned. The oil is very slightly soluble in water and mixes rather easily. Violent agitation, as in a Pachuca tank or in the Minerals Separation type of machine, is often enough to give good distribution and a good flotation froth.

The froth produced by fuel oil is of the same appearance as that of coal-tar creosote as far as the bubbles are concerned, which are nearly the same size, ranging from 2 in. to 4 and 6 in. diameter. The froth, however, is very brittle when the right amount of oil is used and is much cleaner than the creosote. In fact, the froth is almost too clean, so that a clean tailing is hard to make with fuel oil alone. Mixed with some oil like creosote, however, it sometimes does good work. The concentrates from the preliminary treatment often need no further treatment. Where a final product is made as concentrate the tailings are usually re-treated, and in such cases the creosote effects seem to carry through to the other treatment, while the fuel oil often has to be added in the second treatment. Various ores differ slightly on this point. Again, a little fuel oil in a coal-tar creosote mixture will sometimes aid in cleaning the froth.

When fuel oil is in excess, the froth produced is stringy and the bubbles often get to be of such size as to stretch across the machine, sometimes over 2 ft. in length. The froth under these conditions is also dirty and will pass through a cleaner without improving the grade of concentrate. Acid is sometimes

an aid in the use of fuel oil, especially when the oil is in excess. The froth from excess fuel oil is hard to break down.

Pine-tar oil is coming into use more now as a possible flotation agent for copper carbonates, but has disadvantages. It is stiff and flows poorly, so that it must be mixed with some other oils in order to make it of such viscosity as to flow. The froth resembles that from wood creosote and pine oil in appearance of the bubbles, but in texture it is much more fine-grained than the wood creosote. It is a powerful frothing agent similar to pine oil. It is easy to flood the plant with froth from this oil. The froth is characterized by being dirty, but, like pine oil, is capable of holding up considerable coarse mineral as well as gangue. When a flood of this froth reaches the cleaner, it is subjected to very little cleaning owing to its frothing character. Often the addition of a small amount of fuel oil will tend to make a cleaner froth, but with the pine-tar oil a very clean froth must not be expected except at the cost of high tailings.

Oleic acid is used at several plants in this country and is confined mostly to zinc ores. It produces large bubbles and a tough froth. The size of the bubbles varies with the amount of oil used, but ranging from several inches to a foot in diameter. The froth is very persistent and stringy, and the bubble film is often strong enough to support light objects.

Introduction of Foreign Oils.—Under this head are also included greases, even though they are not literally oils. Under no circumstances should these oils or greases be permitted to come into the feed delivered to the flotation machines. A common place to find some of this trouble is around the bearings and gears of the pebble and ball mills; here an enormous lot of grease and oil is used and a great deal wasted. Dripping bearings give their share of trouble. If foreign oils get into the feed before entering the mills, it is then mixed and its effect is distributed over a large amount and does not give quite the trouble as when it finds its way in after leaving the mills. Sometimes acid will remedy the effect of foreign oils when they cannot be avoided in any other way, if their source is ahead of the grinding mills, but the effect of the acid on the flotation must also be considered. Usually the best thing to do is to find the source of oil and eliminate it. Froth from lubricating oils is tough and stringy and carries little mineral value and is almost uncleanable. The bubbles are hard to break down and, when

once broken down, will form again at the slightest opportunity. A large amount of foreign oils come from cleaning up around the plant. The mineral content of tailings is not reduced by the action of these oils; they merely act as an annoying frothing agent that may bury the plant in froth.

As a general rule the best froth to be used in one in which the bubbles are of a medium size and rather tough, although too tough a bubble should be avoided. For chalcocite, pine oil works best, but this froth is delicate and a froth of this kind needs constant attention, so that an operator cannot care for as many machines as he otherwise might. Coal-tar mixtures of from 50 to 80% coal tar and 50 to 20% creosote, with small amounts of pine oil added from time to time, will often give the same results and much steadier conditions than the pine oil on chalcocite ores. The best mixture would have to be worked out in the plant.

Chalcopyrite responds readily to the same mixture as chalcocite, but the pine-oil froth does not do very well with the chalcopyrite. Zinc and lead ores vary greatly. Some will respond beautifully to mixtures of coal tar and wood creosote and some with coal-tar creosote, while others will not respond with any oils but mixtures of wood-distillation products. Then there are still others that seem to require the addition of acid in order to do anything at all.

Further Experiments on Oils.—The question of the determination of the qualities of oils that makes them available for flotation is a serious one, but no way of finding those qualities and testing for them before using the oil has been brought forth. A tentative line of experimentation has been suggested by C. Y. Clayton and C. E. Peterson,¹ of the Missouri School of Mines and Metallurgy, who suggest the investigation of the following qualities: Dielectric strength; optical properties—the power to absorb certain rays; insulation value; viscosity; water-soluble content; facility of emulsification (mixing); surface tension; specific gravity and chemical composition.

Among the experiments suggested are those determining the relation, if there is any, between the selective power or frothing power of an oil and its dielectric constant, and its viscosity. Also to determine whether surface tension and film strength play any part in the frothing or selective power of an oil; whether saponi-

¹ *Min. and Sci. Press*, Apr. 22, 1916.

fication bears any relation to the flotation power of an oil, either as to its frothing or selective value. The facility of rapid subdivision and flotation value. The water-soluble content and its value to flotation. What the quantitative effect of frothing and selecting oils may be, and the possible relation between a mineral's dielectric constant and its tendency to be floated. The grouping of oils as to their chemical composition should be worked out, ascribing flotative value to the groups and comparing the group members. The effect of salts of different kinds in different amounts on flotation oils should be considered, as should also the method of union of oils and bubbles. Many of these points have already been investigated, but others of them have not been, and any additional work would be a benefit to the operation of the process.

Quantity of Oil.—The question of what determines the amount of oil necessary for flotation has been discussed by Oliver C. Ralson in a pamphlet issued by the Bureau of Mines in June, 1917. He points out that the common flotation plants obtain success only through the use of oily flotation agents. As many of these are expensive, the question of use of a minimum amount of them is of considerable importance. Although some flotation plants are now using large amounts of oil, most of them are doing so to avoid paying royalties to the company that owns the patent covering the use of less than 1 % of oil. Some plants have successfully floated galena ore with only one-fifth of a pound of pine oil per ton of ore, although the upper limit is 20 lb. per ton. Wherever the aim is economy in the use of oil, the operators daily cut down the percentage of oil used until the losses of valuable minerals obtaining from using any less are greater than the value of the oil saved.

It is possible that the minimum amount of oil necessary for froth flotation depends upon three different factors—the area of the surface of mineral sulphide exposed for oiling, the amount of water used per ton of ore, and the number of and total surface of air bubbles passing through a given amount of pulp. The question is, then, how to alter these factors in order to obtain economy of oil.

There seems to be some doubt whether the amount of oil necessary depends upon the amount of sulphides exposed per unit weight of pulp. For example, for two zinc ores of about the same screen analysis, about the same amount of pine oil

is necessary for separation, although one ore contains about 20% of zinc and the other only 5%. Many illustrations might be cited of the proportions of sulphide minerals in the pulp varying even more widely while the proportion of flotation oils used was practically identical. It is necessary to consider, in this connection, the exact function of the oil before the question can be discussed.

The ordinary classification of oils is as frothers and oilers, although some few oils possess the properties of each. As the froth consists largely of water and is formed by contaminating water with a frothing agent, it would seem that a certain concentration of the frothing agent would cause a definite kind of froth. Consequently the proper proportions of frothing oil for good flotation should depend on the proportion of water in the pulp rather than the proportion of sulphides in the pulp. However, a collecting oil is usually not mixable with water and, as shown by many experiments, it oils the surface of the sulphide particles so that there is a greater concentration of oil in the flotation concentrates than in the tailings. This evidence indicates that the proportion of the collecting oil required varies with the total area of the exposed surfaces of the sulphide particles. Either more sulphides in the feed or a finely ground feed causes a greater area of surface to be exposed to the water, and hence requires a larger amount of collecting oil.

By considering these points, an intelligent choice of oil mixtures can be made for experiments. For a very slimy ore, the required amount of collecting oils necessarily will be rather high and the amount of a frothing constituent may be low. For granular pulp, a low proportion of collecting and a higher proportion of frothing oil will be needed. Slimes also require a higher dilution of pulp than sands for successful treatment. Such dilution means more tons of water for every ton of ore, and, as there must be a certain concentration of the frothing oil in the water, the use of more frothing oil per ton of ore.

As regards the effect of air supply on the amount of oil used, it is commonly known that the use of a very small proportion of oil and intense aeration often enables a separation of galena and sphalerite. Barely enough oil to float the galena is used, and the large volume of air used makes up for the lack of oil. The films of the air bubbles adhere collectively to the sulphide particles, although aeration of a pulp free from a froth-

producing substance usually gives a weak froth that breaks up too fast to take advantage of the tendency of the air bubbles to lift the sulphide particles.

Apparently then, the increasing amount of air bubbles passing through a pulp permit the use of less oil. Generally the oils used for differential flotation have a good frothing and collecting quality. Passing a large volume of air through a pulp by causing many bubbles, greatly increases the area of the film surfaces separating air and water. As the oil tends to reduce the surface tension of the water by concentrating in the surface film, it is probable that frothing oil would tend to collect in the froth rather than in bulk water. Then, if the froth were raked off, not enough oil might be left to cause good flotation of some remaining sulphides. Such a condition would require the addition of more oil and thus intense aeration would make more oil necessary. In any event, enough collecting oil to wet the sulphides would have to be used.

Alkaline solutions cause better dissemination of most oils and thus make them more effective in oiling the minerals.

These conditions indicate that the rules for obtaining the minimum expenditure of oil are, first, to use a thick pulp and second, to determine the proper ratio of collecting oil to frothing oil by experiment. The more dilute the pulp, the more frothing constituent required, and the finer the mineral, the more collecting oil necessary. Third, use the greatest aeration possible with a given amount of power, and fourth, if possible, use an alkaline pulp.

CHAPTER V

FLOTATION PROCESSES AND APPARATUS

The Elmore vacuum process, the invention of F. E. Elmore, is based primarily upon the fact that, in a flowing pulp of crushed ore and water, oil has a selective action for the metallic mineral particles as distinct from the rocky particles or gangue. This selective action is materially increased, in some cases, by the use of an acid. Secondly, dependence is upon the fact that the air or gas dissolved in water is liberated, partially or entirely, upon subjecting it to a pressure less than that of the surrounding atmosphere. The liberated gases may be augmented by the generation of gases in the pulp or by their introduction from an external source. The gases attach themselves to the greased mineral particles, and being largely increased in volume as a result of the partial vacuum applied, cause the greased particles with their attendant bubbles of air or gas to float to the surface of the liquid. The process has been extensively reviewed by Alexander Stanley Elmore,¹ from whose work this description is taken. The operation of the process is shown by reference to the accompanying drawing. The pulp containing crushed ore flows continually into the mixer into which also is introduced small quantities of oil and, if required, of acid also at the point *B*. The agitation is secured by rotation of the beater. The agitated pulp flows continuously from the mixer into the funnel. The concentrate-discharge pipe *E* and the tailing-discharge pipe *F* are both sealed with water in the tanks.

The upper end of the feed pipe enters the center of a conical separating vessel *I*. The rate of flow of the pulp down the pipe *F* being slightly less than the inflow of the feed pipe *D*, a small amount of liquid overflows the rim of the annular space *K*, this quantity of liquid being sufficient to carry the concentrate in the pipe into the tank. The rakes *L* are caused to rotate slowly by means of the worm and wheel *M*, the angle of the blade being such as to cause the solid matter in the pulp to

¹ *Eng. and Min. Journal*, May 11, 1907.

travel from the center to the outside of the conical chamber, whence the tailings continuously discharge down the pipe *F*. The feed pipe is usually about 25 to 30 ft. long, the tailing and concentrate pipes *E* and *F* being a few feet longer, so that in effect the feed pipe and tailing pipe form the long and short legs of a siphon; thus, the power required to elevate the pulp into the conical chamber is supplied by the falling of pulp in

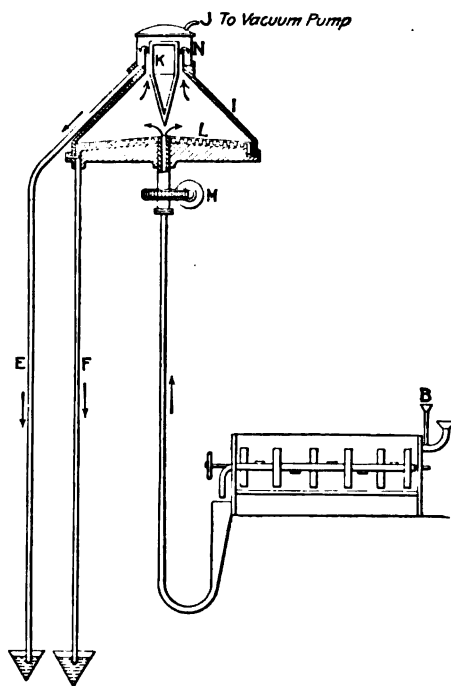


FIG. 6.—The Elmore vacuum apparatus.

the tailing pipe. So long as a continuous flow of pulp is supplied to the mixer, a continuous and entirely automatic discharge of tailing and concentrate is secured. The annular space *K* is surrounded by a thick glass cylinder, or by a metal cylinder with one or more thick glass windows through which the discharge of concentrate over the edge of the annular ring may be observed.

The apparatus is constructed of metal with the exception of the glass windows, the mixer being a simple wooden trough with wooden beaters. The power required for a 5-ft. unit of such a

plant does not exceed 2 to $2\frac{1}{2}$ hp., including that required for driving the vacuum pump, mixer, and separator. The apparatus is usually constructed in standard units, the conical separating chamber of which is 5 ft. in diameter.

The capacity of the apparatus naturally varies with the kind of ore to be treated, the fineness to which the ore has to be ground, and some other conditions, but a 5-ft. machine will ordinarily treat from 35 to 45 tons of crude ore per 24 hours. The cost of the plant is low for so large a capacity. Extensive foundations are not required, because the beaters in the mixing cylinder only make from 30 to 40 r.p.m., and the rakes in the conical separating chamber only 1 or 2 r.p.m. Vibration is therefore entirely absent. No sizing of the pulp is required and the operation of the plant is entirely automatic.

The quantity of oil and acid required is not the same for all ores, but usually amounts to not more than a few pounds, say from 3 to 10 lb. per ton of ore treated. Probably the following figures will be of interest as giving a fair idea of working costs. Naturally, the price of acid, oil, and labor will vary at different mines: Oil, say 7 lb. per ton of ore treated at 1c. 7c.; acid, where required, 6 lb. per ton at 1c., 6c.; labor at \$2.50 per 8 hr., 11c.; total 24c. This is on the basis of a plant of, say, 200 tons per 24 hr. capacity. To the above, the cost of, say, 0.05 hp. per ton has to be added, say 10 hp. for a 200-ton plant.

The Macquisten Tube Apparatus.—The Macquisten process has already been described in general, but it would be well to consider further its action in practical application. The process has not been widely applied, at any rate in the United States, but at least one important plant still exists—that of the Morning mill of the Federal Mining and Smelting Co., at Mullan, Ida. The Macquisten-tube installation at this plant has been recently increased. Another installation is at the Iron Mountain mill, in Montana, just over the border from Idaho.

The best description of the Macquisten process was given by W. R. Ingalls,¹ who described its installation at the Adelaide Reduction Works, of Golconda, Nev. This description is complete and exceptionally clear, so that exhaustive quotations from it will be the best method of illustrating the practical working of the process.

¹ "Concentration Upside Down," *Eng. and Min. Journal*, Oct. 26, 1907.

The Adelaide Reduction Works was erected a number of years ago to reduce the ore of mines that had been acquired by a mining company in the vicinity of Golconda, chiefly at Adelaide, about 14 miles south of Golconda. The works comprise a concentrating mill and a small smeltery. The latter has two Bruckner roasting furnaces and three small reverberatory furnaces, rated at capacity for smelting about 30 tons of ore per day per furnace. The works were operated for a brief period, but were found unable to treat successfully the Adelaide ore, and consequently were closed, since when they have been run only when some new process was to be tried.

The Adelaide ore is chalcopyrite, with a smaller proportion of pyrrhotite and pyrite, and a little blende and galena. These minerals are rather finely disseminated in a hard, dense, quartzose gangue that also contains spinel and garnet. The presence of the heavy gangue minerals caused concentration by ordinary treating and tables to be ineffective. Wilfleys, vanners, and other forms of tables all failed. Then a trial was made with the Blake-Morscher electrostatic separator, which is reported to have given fairly good results, but not good enough to induce its adoption. The company installed a trial plant of Macquisten tubes which solved the problem of economical metallurgy. The interesting part of the process begins with the separating tubes.

Construction of the Tubes.—The Macquisten tubes are made of cast iron. They are 6 ft. long, 1 ft. in diameter inside, and weigh about 450 lb. each. Externally they are cast with two tires which rest upon the supporting rollers. On the latter the tubes rest horizontally. The discharge end of the tube is entirely open and the feed end is closed, except for a hole in the center large enough to admit the pipe that feeds the pulp. Internally the tube is cast with a helical groove, the form and dimensions of which are shown in the accompanying sketches. The tubes in use at Golconda at the time of operation had an exterior helix of $\frac{3}{4}$ -in. pitch, but it was intended to substitute for them tubes of 1.5 in. pitch, which is a later and improved form that has been found to give better results. The discharge end of the tube connects with a vessel called the separating box, the joint between the latter and the tube being watertight while the tube is free to revolve. At the side of the separating box, directly opposite to the discharge end of the tube, an opening is cut out for the overflow of the surface layer of water carrying the float-

ing mineral. This opening, or weir, regulates the depth of water in the tube. At Golconda the bottom of the weir is 3 in. above the bottom of the tube, inside; consequently there is 3 in. of water in the tube. The feed of pulp to the tube and the discharge of the tailings from the bottom of the separating box are so regulated that the water passing over the weir is about $\frac{1}{32}$ in. deep. The tailings flow out through a spigot in the bottom of the separating box, carrying them to a second tube for further treatment, or to the waste dump, as the case may be.

In operation the tube is rotated at 30 r.p.m. in a direction corresponding to the helix of the interior. The pulp is thus

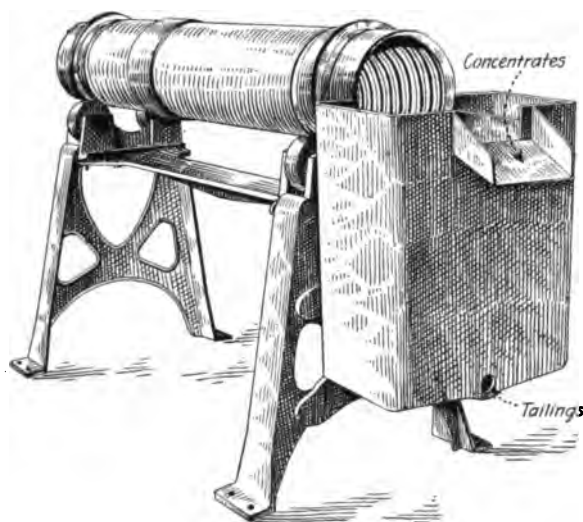


FIG. 7.—The Macquisten tube.

screwed through the tube and in its advance is repeatedly given an opportunity to slide upon the surface of the water where it may be retained by surface tension. Considering the pulp lying in any single corrugation; as the tube revolves the pulp is carried up to the surface of the water, some of it to a little above the surface, and gently rolled over. The angle of emergence is such that some particles of sulphide minerals slide upon the surface of the water, and once that is accomplished they appear to remain on the surface. The gangue minerals simply slide back into the water. This action is taking place in every corrugation, wherefore the minerals that are capable of being

upheld by surface tension are given many chances to slide back upon the water at just the right angle. With four tubes in series some of the particles can be observed taking advantage of the chance to float at the very end of the fourth tube. However, the maximum floating occurs in the first tube, and that which takes place in the fourth is comparatively insignificant and is confined chiefly to the larger and consequently more difficultly floatable particles of minerals. It is believed that with the improved tubes, two in series, each 6 ft. long, will be all that is commercially required, and judging from the work at Golconda, this opinion would appear to be correct.

In any given tube revolving clockwise (looking toward the feed from the discharge end) the flotation takes place chiefly at the left-hand side, where the pulp emerges from the water. The bulk of the pulp in any corrugation either slides back there into the water or floats off; however, a certain amount always adheres to the tube and is carried over to the water on the right-hand side where, upon meeting the water, a further flotation takes place, but naturally this is much less than on the left-hand side. Consequently, from the time the pulp enters the first tube there are floating sulphides streaming off from the sides of the tube, which by the time the discharge end is reached form a glistening film covering the entire surface of the water both in the tube and in the separating box. This is quite different in appearance from the black, frothy scum that rises to the surface in acid flotation processes. In separating the chalcopyrite and pyrite at Golconda the surface of the water is covered with a sparkling froth and mineral, the water looking much as if it had been sprinkled with the bronze powder that is employed for decorative purposes. In treating a lead ore there comes off a film of steel gray galena; in treating a zinc ore from Joplin, Mo., there was a film of mustard-yellow blende. The flotation is persistent. Once it has been effected, the mineral remains on the water until the latter has overflowed into the launder, which spouts it into the concentrates collecting tank, and even in the latter more or less mineral can be observed floating on the surface.

Results Obtained by the Tubes.—As to the technical results at Golconda, the capacity of a single tube is placed at 5 tons of ore per 24 hr.; consequently the capacity of the mill with its present arrangement is 125 tons per 24 hours. The ore of the Adelaide mine is reported to assay 2.7 to 3.2% copper, but the mill

was running on dump stuff that was said to assay only 2.2% copper. Having been ground to 30-mesh size this was concentrated in a ratio of 11:1, yielding concentrates assaying 22% copper, and tailings assaying about 0.2%. This indicates an extraction of a little more than 90%, but refers only to the deslimed ore. The slime was run into a settling pond and accumulated pending the determination of plans for its further treatment. This slime was believed to amount to 30% of the original ore and to be of about the same assay in copper as the latter. Consequently the actual extraction was about 0.10 to 0.9. It was 63%, but it was considered the percentage of slime could be reduced, and of course the percentage of extraction by the tubes will increase directly in proportion as the percentage of slime is reduced.

Like almost every process, this one has its drawback, and as in the case of almost all processes of ore dressing, the drawback in this one is the treatment of the very fine material that in the dry processes is dust and in the wet processes is slime. It is obvious that this new process, wherein the concentrate is floated off on the surface of the water in the most liberal sense of that expression, it will be contaminated by fine slime that does not settle quickly and floats just under the surface of the water. It would appear that this difficulty might be greatly ameliorated by a suitable modification in the form of a settling box, possibly increasing the length of the latter, reducing the velocity of the water passing through it, or otherwise giving the fine slime an opportunity to settle $\frac{1}{20}$ of an inch or so below the surface before reaching the weir.

The foregoing description is complete and clear, and does not require additional explanations, which could, at best, only duplicate what Mr. Ingalls has already so admirably explained.

The Wood Surface-Tension Process.—The principle of the Wood process for the concentration of ores is the utilization of the surface tension of water, either fresh, acid, or salt. The machine for utilizing the process was described by Henry E. Wood in 1912 but since that time has been materially improved. Its application to molybdenite concentration was described by Wood in a paper read before the Canadian Mining Institute, at Ottawa, in March, 1916, from which the following description is taken.

The machine consists of a rectangular structural-steel frame

about $5\frac{1}{2}$ ft. high at the back, 5 ft. wide, and 5 ft. long. Supported in and on this frame are a series of feed hoppers and tanks, feed mechanisms, etc., as will be described in detail, and shown in the accompanying drawing. At the highest part, which is the back side of the frame, is supported a rectangular feed hopper made with cast-iron ends and sheet-steel sides and bottom. Front, ends, and part of the back are vertical. About two-thirds of the back slopes to the front at an angle of 45° . The bottom of the feed hopper is about 4 in. wide and 4 ft. long, and is made of a removable steel sheet that is used as one of the feed-opening adjustments. On the front of the hopper is a vertical movable gate which, in combination with the bottom, gives a wider range of adjustment of the feed openings. Inside the hopper is a longitudinal shaft studded with short pins, which is revolved at the rate of 80 to 100 r.p.m., and serves to keep the ore in the hopper loose and free from channels.

Suspended from the upper frame members by means of flat hickory strips is a triangular truss with cast-iron ends. This truss carries the feed bar which is made of angle-iron and wood, carrying a number of $3\frac{1}{2} \times \frac{1}{4}$ in. screws which project into the feed openings of the hopper. Below the feed bar in the truss is supported an adjustable feed pan 8 in. wide and 4 ft. long that receives ore from the feed hopper as it is delivered by the feed bar. The truss carrying the feed bar and feed pan is vibrated in a horizontal plane by means of a small eccentric attached to the main frame of the machine and connected to the truss by means of a flexible hickory drive rod. The eccentric shaft is driven at about 500 r.p.m. The ore dropping at the back of the feed plate is spread out into a thin, even sheet by the time it has traveled 8 in. at right angles to the vibrations, and is then dropped on to the surface of the water that is contained in the main tank, which is a rectangular inverted pyramid.

The main tank is supported in a frame at ends and back, has a valve at the point or bottom, above which is a diaphragm to steady the flow of tailing that collects in the tank. At the front of the tank is a partition that has its top edge $\frac{3}{4}$ in. below the top of the tank. This edge is an overflow weir and maintains a constant level of water in the tank. At the back of the tank is supported a spray pipe which discharges a large number of fine jets of water parallel to and just below the surface of the water contained in the tank, thereby creating a swift and smooth sur-

face current toward the front of the tank. Halfway between the back spray pipe and the front of the tank is a second spray pipe with a smaller number of holes that assists the back pipe to maintain the surface current uniform clear across the tank. In front of and bolted to the main tank is a narrow rectangular cone that has a bottom discharge and an overflow weir as in the main

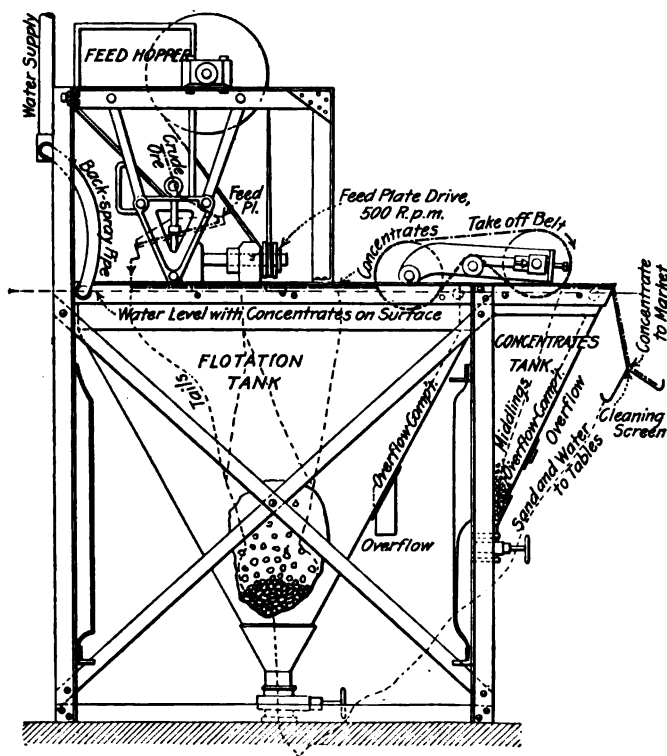


FIG. 8.—The Wood surface-tension machine.

tank. The joint between the tanks is made watertight by means of light iron channels set over the top of the joint. Supported on top of the main frame directly over the joint of the tanks is a take-off or concentrate-collecting device, which consists of two 6 in. by 5 ft. rollers and one 2½ in. by 5 ft. roller, supported at the ends of adjustable arms. Around these three rollers is passed a heavy fabric belt that picks up the concentrate from the surface of the water in the main tank, carries it over to the

concentrate tank where it is washed off on to the surface of the water in that tank and passed on over the weir to a settling or filtering device. The main shaft of the machine is belt-driven and may be connected with any suitable source of power. The roller in the feed hopper and the take-off rollers are driven by chain and sprocket gearings. The eccentric shaft for vibrating the truss is driven by a small round leather belt. The power required is about $\frac{1}{4}$ hp., and the water required varies from 5 to 10 gal. per minute. A partially submerged corrugated rubber cylinder is sometimes substituted for the moving plane of water.

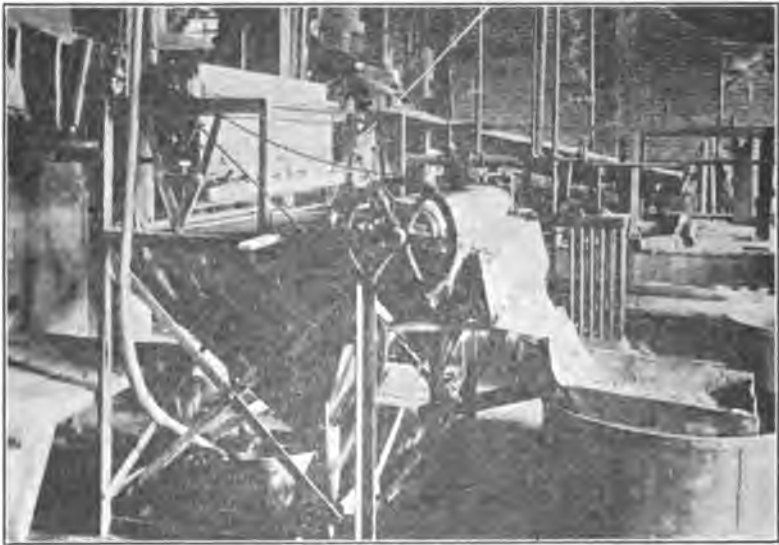


FIG. 9. --A Wood installation.

Operation of the Wood Machine.—In operation, the dry crushed ore, sized or unsized, falls from the hopper to an inclined vibrating plate and thence to the surface of moving water in the tank. Adjustments can be made permitting the ore to fall from a minimum of $\frac{1}{4}$ in., up to a maximum of 4 in., depending upon the size of the material. A -10-mesh ore would all penetrate the surface film if dropped 4 in., while in falling only $\frac{1}{4}$ in. in the molybdenite will float upon the water and the gangue will sink. A -30 or -40-mesh ore requires a greater fall, so that the individual particles will meet the water surface independently. If the minimum drop is used with such fine

material, much of the gangue would land on floating portions of the mineral, and not having a chance to be wetted, would contaminate the concentrate. Jets of water under control keep the surface of the water in motion across the tank toward the take-off belt, so that an area of fresh-water surface is always ready for the falling feed. While passing over the surface of the tank, most of the gangue is wetted and sinks. The molybdenite film adheres to the emerging belt, is lifted out of the tank, and at the point where the belt reverses its traveling direction, it meets the level of water in a smaller tank and leaves the belt. Crossing a

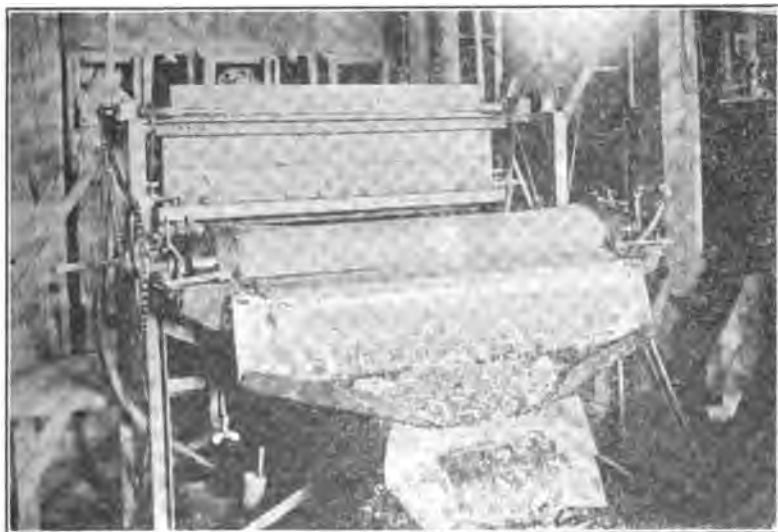


FIG. 10.— Another view of a Wood installation.

few inches over this second tank, it falls upon a nearly vertical dewatering screen of the same mesh originally used in crushing the ore.

The result of this treatment is that the film of molybdenite is taken away from the greater portion of the gangue, which either sinks or is suspended in the main tank. Such particles of partially wetted gangue as are forced on to the belt become saturated en route and are dropped into the second tank as a middling product. This is often as clean as the other tailing, but if molybdenite is there to any appreciable extent, it is reserved for retreatment. Particles of gangue and gangue in suspension, that are not dropped in the second tank, readily pass through

the meshes of the inclined screen and also constitute a middling product. Both of these middlings are usually of small proportions by weight and should be considered as part of the material to be cleaned by retreatment. In some cases the additional wetting of the gangue, as well as drying the exposed sulphide, is so marked that canvas-covered cylinders are placed in the main tank, the film being lifted up twice and deposited twice before it arrives at the main take-off belt.

The adoption of this plan produces such clean concentrates that it is not necessary to treat again the main product, as has to be done to raise the grade. The molybdenite is diverted to suitable concentration boxes, dewatered by decantation or filtering, and after drying is ready to be sampled, assayed and shipped. Some of the smaller tests reported herewith were made from ore crushed at - 10, - 12, - 16, and -20-mesh, and when these reach a 30- or 40-mesh final-cleaning screen particles of gangue are too large to pass through it, being collected with the concentrate and thereby lowering the grade of the latter. The small machine used for the preliminary investigation has a 10-in. exposed belt surface as against 54 in. in width, of the commercial concentrator. It also has but about one-half of the length of the flotation period provided in the large machine. As a result of these two factors, a commercial machine usually gives better result than a small laboratory device. The gangue and any associated mineral, not so readily susceptible of flotation as molybdenite, passes through the film and becomes dependent upon gravity conditions for separation. In the preparation of the ore, molybdenite flakes may have lost their original tabular shape, or for other reasons be not adaptable to flotation. On this account, the tailing after flotation passes to a Wilfley table. A concentrate is obtained that carries the coarse molybdenite as well as other minerals of value such as copper, free gold, tungsten, etc. The Wilfley concentrate is subject to retreatment for additional molybdenite as well as the other minerals mentioned. Satisfactory adjustment having been made, the operation of the machine then becomes automatic and requires only the usual attention given to milling operations.

The Capacity of the Machine.—The capacity of the machine is mainly dependent upon two conditions, that is size of ore treated and the ratio of concentration. The surface area of the water in the tank can sustain only a definite weight of concen-

trate, and to overload it will cause the mineral to sink. The water film is elastic and the concentrate very compressible, so that it will sustain a much larger weight than seems possible. A sheet of water 38×38 in. will support 1.55 lb. of molybdenite. From this it is evident that excess feeding and overloading must be avoided. To increase the capacity as effected by such conditions would necessitate a greater area of tank surface, so that with other parts of the machine in proportion, a greater amount of ore could be milled per machine. It is merely a question of how wide a feed plate and take-off belt can be safely operated. A 0.5 to 1% molybdenite ore with a gangue that is quickly wetted, such as clean quartz, will pass through very rapidly when crushed to ordinary sizes. The feed may be sized or unsized, but if a sized feed is given, it will pass through at a much greater rate. The machine now in use, with feed and take-off belt of 52-in. width will treat a 1% molybdenite ore at a rate varying from 10 to 20 tons per 24 hours. A 10% to 20% or better ore will go through at a much less rate, in proportion to the tank area. Other minerals frequently float and take up just as much space as molybdenite, as for instance chalcopyrite. When it is present it requires a second treatment, when the partial oxidation caused by the intermediate heating and drying will cause it to sink, while the molybdenite which does not oxidize so readily, will float. Under usual conditions, the capacity of the machine will vary from 4 to 15 tons per day.

The Minerals Separation Process.—The system of flotation now utilized by Minerals Separation is a development of the oil-and-air-froth system by means of which a mixture of pulp, oil, air, and sometimes acid is violently agitated, producing a froth that floats and carries the mineral particles along with it. The machine now principally in use is a development of many workers with many ideas, but possibly the original Hoover machine had more to do with the present result than any other. The accompanying sketch shows the essential principle of the machine, a development of the original machines.

At *F* is a mixing compartment containing an agitator running at high speed; *Z* is the spitzkasten. The ore, water, oil, acid, and sometimes steam, if it is required for raising the temperature of the pulp, are fed into the mixing compartment in proper proportions where the pulp is thoroughly mixed and violently agitated. *F* has a hole in the bottom which allows the

pulp to pass into the next compartment where there is more violent agitation. The pulp passes from this into another agitation compartment and then goes over a weir into the spitzkasten where a heavy, dense mineral froth is floated over the lip. The gangue along with some of the sulphides, passes to the tip of the spitzkasten Z, the tip being connected by a pipe with the bottom of another agitation compartment. The agitator in this compartment acts as a centrifugal pump and draws the material up into the compartment where the pulp receives further agitation and

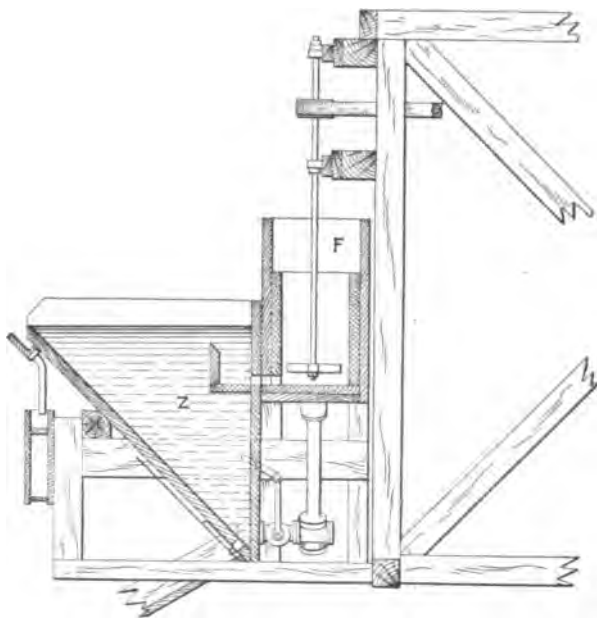


FIG. 11.—A section of the Hoover single-level apparatus.

aeration. The operation of the machine then is simply a repetition of agitation and skimming from the spitzkasten.

Present Minerals Separation Machine.—The Minerals Separation machine as used at the present time is somewhat different. A description of the installation at the Washoe Reduction Works of the Anaconda Copper Mining Co. will illustrate the construction of the modern machine and its operation. The accompanying drawing shows the machine as it is installed. The feed is introduced into the first agitator box at the motor end of the machine. From this box it passes to a second box through an

opening in the partition.—shown on the elevation drawing. From this second agitating compartment, the pulp passes into the first spitzkasten where the first concentrate froth is removed by means of a paddle and the remaining pulp passes through a pipe, the inlet to which is controlled by a valve, to the third agitating box. From this agitating box the pulp passes to the second spitzkasten, and so on down through the machine until the pulp is introduced into the fourteenth spitzkasten. The discharge from the fourteenth or last spitzkasten leaves the machine as tailing. The first four to seven spitzkastens make finished con-

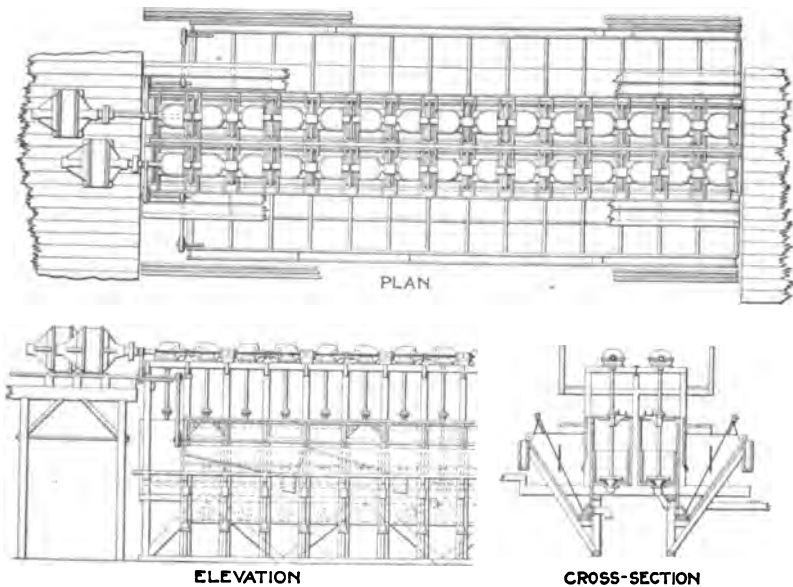


Fig. 12.—The Minerals Separation machine at Anaconda.

centrates and the remaining spitzkastens make middlings that are returned to the system. The drawing shows a double machine, each machine, however, being run as a separate unit. The line shaft is driven by a 150-hp. motor at a speed of 385 r.p.m. The machine under a load requires about 100 hp. The vertical shaft carrying the impellers revolves at a speed of 223 r.p.m. The impellers are the usual Minerals Separation type, having four blades placed at 45° to the vertical. They are 2 ft. in diameter. Each pair of impellers is arranged so that one revolves in one direction and the other in the opposite direction. This

tends to balance the side thrust on the line shaft. The bevel gears that drive the impeller shaft are cut steel and run in grease. The impeller shaft is supported both vertically and horizontally by ball bearings. In some cases it has been found of advantage to place the motor on the other end of the machine from that shown in the drawing, especially if it is necessary to heat the pulp to over 100°F. There is more or less steam arising from the pulp, especially at the feed end, which condenses on the



FIG. 13.—A flotation installation of the Minerals Separation type at a Colorado tungsten mill.

motors and by placing the motor at the discharge end this trouble is practically eliminated.

Each machine makes three products, concentrate, which goes to the dewatering division; middling, returned to the head of the machine, and tailing, to waste. Concentrate is taken from the first three to five spitzkastens and middlings from the last nine to eleven. A portion of the pulp is overflowed from the last three spitzkastens, together with the froth. About 6 to 8 lb. of 50° Baumé sulphuric acid per ton of flotation feed is used,

together with 2 to 3 lb. of kerosene sludge acid and $\frac{1}{2}$ to 1 lb. of crude wood creosote.

A portion of the wood creosote is added ahead of the Hardinge mill (about 0.03 to 0.05 lb. per ton of feed), and the remainder is added in the six agitating compartments. The sulphuric acid and sludge acid are added at the head of the machine. The pulp is heated to from 60° to 70°F. by passing live steam into it at

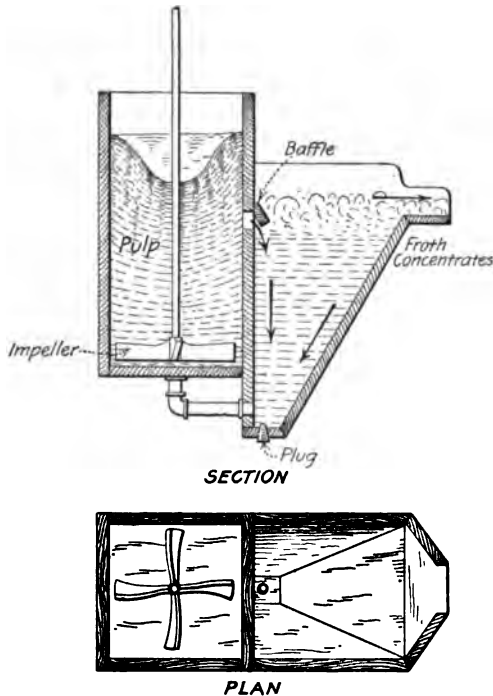


FIG. 14.—A machine of the Hoover type.

the head of the machine. Three machines are used for treating sand, and the fourth for treating current slime from the upper portion of the mill. Each machine has a capacity of about 400 tons per day on sand, and 175 tons per day on slimes.

The plant treating slime consists of 20 Minerals Separation machines of the same type as those used in the concentrator, and five 20 × 12 ft. Dorr tanks are used for dewatering the concentrate. It is noted that the treatment of slime from the dump offers a few more difficulties than that of the current slime. The former has been exposed to the weather for a number of years, so

that the copper is partially oxidized, consequently that portion probably cannot be floated. At Great Falls, one double Minerals Separation machine of the same design and size as used at Anaconda is treating from 300 to 350 tons of dump slimes daily.

Other Minerals Separation Machines.—The machine used by Minerals Separation varies in different plants. Different ideas have been developed by the operators on the staff of Minerals Separation and outside inventors who have transferred their ideas to that company. One of the important modifications is known as the Hebbard machine, one of the late developments of which is in principle similar to the Hoover. The vital difference

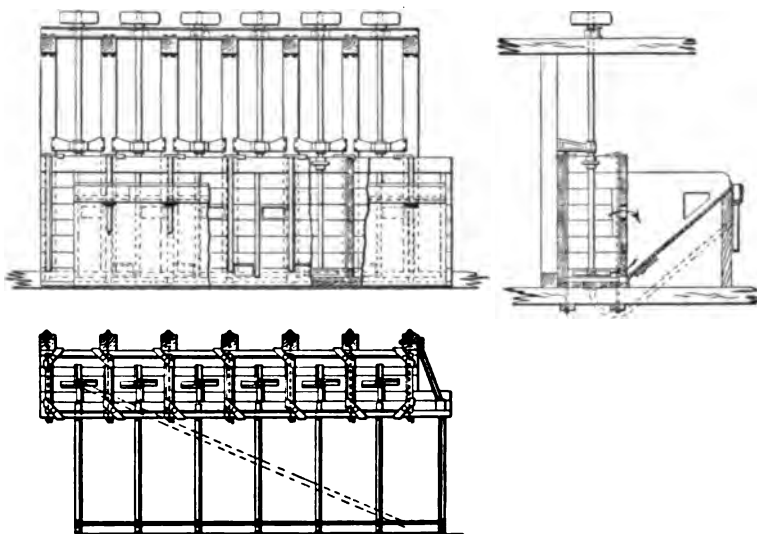


FIG. 15.—Improved Hebbard machine.

between the Hebbard and Hoover types is that in the former dependence is not altogether placed upon the beating of atmospheric air into the mixture while agitation is taking place. On the contrary, a pipe containing air under small pressure is led to a point just beneath the agitating or beating mechanism in the pulp, and a stream of air is delivered to that part. Naturally a far larger quantity of air can be beaten into the pulp in this way than if dependence were placed merely upon the vortex action of the title, as it draws air upon the surrounding atmosphere. The Hebbard machine, like the Hoover, was originally intended to be driven from the top, but the Harvey

improvement is one by means of which the drive is arranged from the bottom, beneath the flotation apparatus. This is said to be a considerable convenience in many cases, and such an arrangement is in practical use at the new zinc plant installed by the Sulphide Corporation at its Central Mine, at Broken Hill, N. S. W.¹ The plant has a capacity of about 4000 tons per week. The boxes, or frothing machines, are fitted with special stirrers to agitate and aerate the pulp, creating a froth containing only

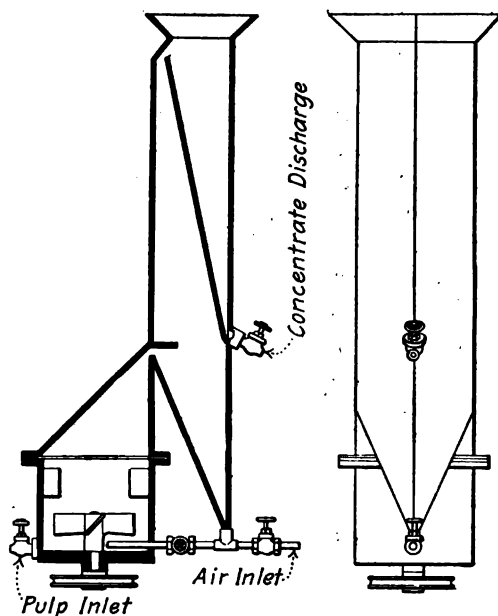


FIG. 16.—The L. A. Wood flotation device.

lead concentrates. Having removed all the lead possible by this means, the pulp passes to other boxes, or frothing apparatus, where sulphuric acid and oil are added. Agitation and aeration take place as before, and the blende is recovered. There are altogether ten boxes, three for lead and seven for zinc.

The Louis A. Wood Machine.—Another machine that has been taken over by Minerals Separation is that of Louis A. Wood, which has been patented. The apparatus is shown in the accompanying drawing. A paddle agitator is provided, the pulp entering from one side and air from another. The whole mixture is

¹ *Min. and Eng. Review*, Dec. 6, 1916.

forced upward and through the small opening in the first chamber, where a middling is formed, which settles back into the air pipe and is forced again into the agitating chamber. The upper chamber receives the clean sulphides, which are then floated nearly to the top of the machine.

Air bubbles containing sulphides and without any oil will not persist to the surface of the pulp, but will break at or below it. This machine is intended to be used without any oil. To receive this sulphide, which sinks from the broken bubbles, the upper chamber with the inclined bottom is devised. The concentrates are drawn off through the valve at the apex of the chamber. Agitation and air injection are at such a rate that the bath gangue will not pass out of the agitating chamber. A special opening near the bottom of the chamber is provided to draw off the tailings. The accompanying drawing shows clearly the construction of the apparatus.

The Callow Bubble-Column Process.—The Callow flotation process is one that depends upon the making of a froth from oil and air, with or without the addition of acid or other chemicals. Instead, however, of producing this froth by means of violent agitation, as is done in most machines, it is produced by a thorough preliminary mixing of the oil and pulp and whatever other ingredients are used, and then the forming of the actual froth in the special Callow device by means of finely divided air bubbles. A description of the process was given by J. M. Callow, its inventor, to the Feb., 1916, meeting of the American Institute of Mining Engineers at New York, and from that description the following explanation is, in great part, taken.

The various elements composing the Callow process are shown in the accompanying drawing. In the mixer, *A*, operated by compressed air, the oil, air, and water are mixed and agitated. In cases where the oil or frothing agent can be fed into a crushing or grinding machine, this mixer or Pachuca tank, can be dispensed with, the tube mill discharging the mixture to the separatory cell.

The initial or roughing separatory cell, *B*, consists of a tank about 9 ft. long and 24 in. wide, with a bottom inclined at from 3 to 4 in. to the foot. It is 20 in. deep at the shallow end and 45 in. deep at the other end. It may be built of either steel or wood, but wood is preferable. The drawing shows the cell in detail. The bottom of the tank consists of a porous medium made of

four thicknesses of loosely woven canvas twill, properly supported by a backing of perforated metal to prevent exploding when under air pressure. Through this porous medium compressed air is forced by a blower. Porous brick or any other ceramic material that will give the necessary fine subdivision to the air, may also be used. Some of these have been tried out, but for practical and mechanical reasons the loosely woven canvas twill seems to serve all purposes better than anything else. The space underneath this porous bottom is divided into eight compartments, each connected by an individual pipe and valve with the main air pipe. By this means the air pressure to each com-

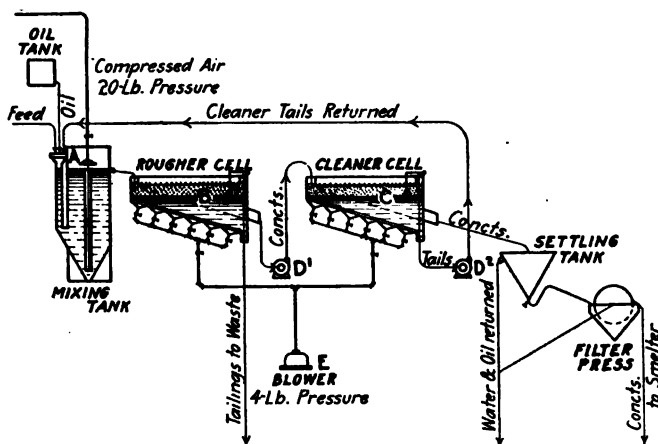


FIG. 17.—Illustration of the Callow process.

partment can be regulated (by throttling the valve) to correspond to the varying hydraulic head in the tank and so as to discharge a uniform amount of air throughout the length of the bottom. A pressure of from 4 to 5 lb. is commonly used, each square foot of porous medium requiring from 8 to 10 cu. ft. of free air per minute.

Each longitudinal edge of the tank is provided with a lid and an overflow gutter for the reception of the froth discharged. The lower end of the tank is provided with a spigot discharge fitted with a plug valve, operated by a plug, to maintain a uniform water level within the tank and thus, in turn maintain a uniform and constant discharge of froth under all the varying conditions of feed supply incident to practical milling operations. The

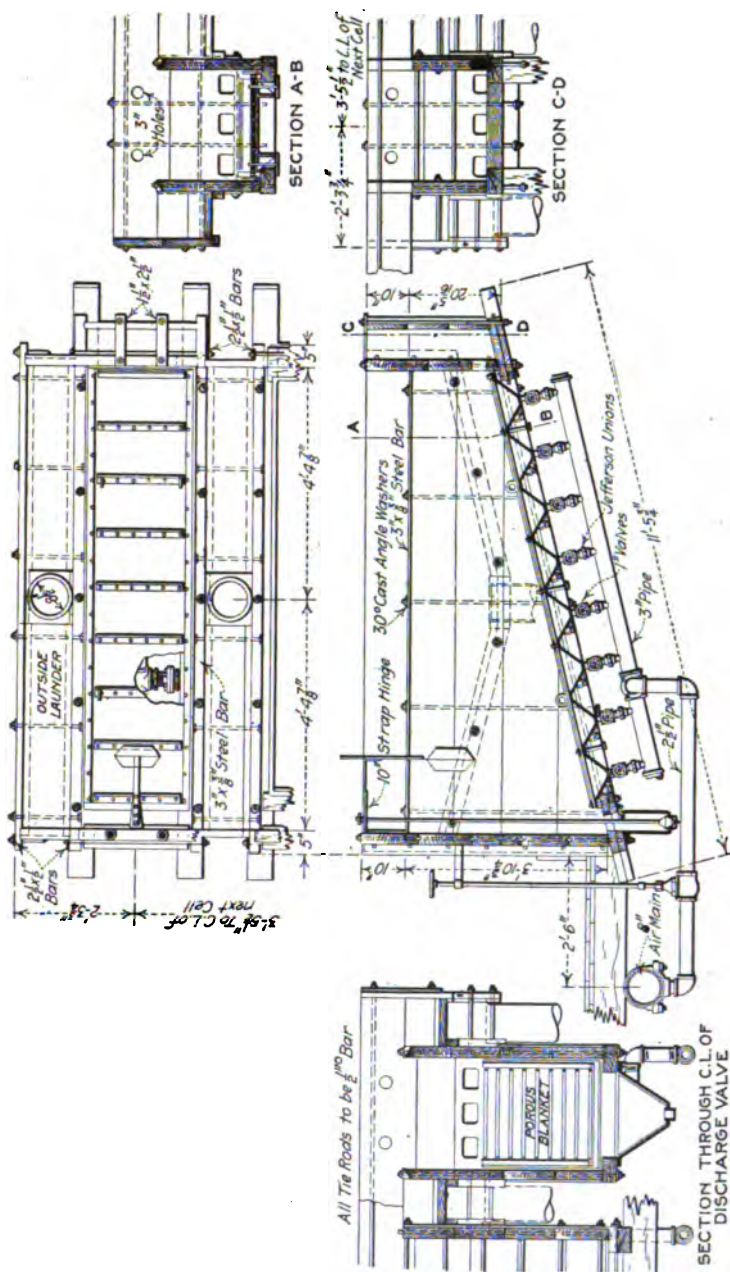


FIG. 18.—The Callow bubble-column apparatus.

water level may, of course, be varied, but is usually maintained at about 10 to 12 in. below the level of the overflow lift. The tailing is discharged through the spigot, and the frothy concentrate is conveyed to the cleaner separatory cell, *C*. This cleaner cell is a machine of the same construction as the rougher. In operation, however, it usually has a lower air pressure. The tailing from the cleaner is returned to the original feed, a closed circuit being thus maintained. The concentrate from the cleaner is the finished product.

The machine may be run either in parallel or in series without any sacrifice in the capacity for a given number of cells. Recent experience goes to show that on some ores the series treatment gives slightly cleaner tailings; on others it does not. It is not necessary to extend this arrangement of cells beyond two cells in series. In a heavily mineralized ore this arrangement is practically advantageous, but in such a case the rougher concentrate might be of high enough grade to dispense with the cleaning operation. The froth from the second cell in the series might be returned to the original feed in the same way that the tailing is returned from the cleaner when practicing a roughing and cleaning operation. A number of such operations and combinations are possible. The froth is generated as a result of injecting the finely divided air into the bottom of the already highly mixed pulp; it continues to form and to overflow as long as it is provided with pulp of the proper consistency, adequately mixed with the right quantity and kind of oil or frothing agent. Measured from the water level within the tank, the froth produced may be from 14 to 16 in. thick, but will be more or less voluminous, coarse or fine grained, dry or watery, according to the character of the ore and the kind and quantity of oil or ore introduced.

The condition of the froth may be varied, therefore, by changes in the kind and quantity of oil used and the quantity of air injected. In some ores rich in sulphide, and where a comparatively low-grade concentrate will suffice, the cleaning cell may not be necessary, but on low-grade ores having a higher ratio of concentration and when a concentrate of extreme cleanliness and of maximum grade is required, a cleaner is desirable.

Density of the Pulp.—The pulp to be treated may be of varying density, from $2\frac{1}{2}$ of water to 1 of ore, up to 5 or 6 to 1. For a mixture of sand and slime, the former ratio is preferable, but for pure, minus 200-mesh slime, the larger proportion of

water is allowable. The particular density is not a matter of so much importance as the supplying of pulp of uniform density, since each variation in the density of pulp requires a readjustment of the oil supply, the quantity of oil increasing in proportion to the increased volume of the pulp, independent of its solid contents.

In the Callow process, as in other flotation systems using oils, the latter may be broadly divided into frothers and collectors. The pine oils are good frothers, and coal tar and its various subdivisions are good oilers. On some ores crude pine tar will in itself combine the properties of both frothing and collecting; on others this may have to be enriched by the addition of some one of its more volatile constituents, such as refined pine oil, turpentine, or wood creosote.

Generally speaking, the coal-tar products are poor frothers, and to get a sufficient volume of froth to insure high recovery, it is often necessary to add refined or crude pine oil, creosote, etc. At the Inspiration mill, where the Callow process is used, the mixture is 80% crude coal tar, and 20% coal-tar creosote; at another plant on similar ore, 45% El Paso coal tar, 40% coal-tar creosote, 10% creosol, and 5% pine oil. At Daly-Judge a mixture of 40% crude coal tar, 40% creosote, and 20% pine oil was used. In the Coeur d'Alene on zinc ores straight wood creosote was used; on the ore of the National Copper Mining Co., plain turpentine will work, but pine oil is better.

Various quantities of oils are used, according to the ores to be treated. In experimental work the stability of bubbles depends to some extent upon the oil used and the nature of the gangue in the pulp treated. The character and characteristics of oils are, however, described in another section, so it will be unnecessary to devote space to that explanation here.

It will be noticed that in the application of the Callow process, the preliminary mixing, or, as it has been wrongly called, "emulsification," is performed in a Pachuca tank. This tank is a device that has been lifted bodily from cyanide practice, where it is used for agitating the slimes in contact with cyanide solutions. After a trial, in practice, however, it has been found that the tank may be omitted with perfectly satisfactory results, the mixing secured in the grinding and conveying sections of the mill, amply meeting all the requirements.

As a side remark, bearing on the wrong use of words that is

too common in our technical literature at the present time, it is proper to object to the use of the word emulsification as synonymous with mixing. The two operations are essentially different, and results obtained after the application of one process may be radically changed if the other is used. However, a discussion of this point occurs in the section devoted to the theory of the process, where the matter is thoroughly thrashed out.

It is to be noted that in the application of the Callow process, the first operation, that of roughing, is devoted to the securing of clean tailings. Since it is almost certain that clean tailings and clean concentrates cannot be obtained in one operation, in the roughing process the grade of concentrate is sacrificed to the cleanness of tailings. Tailing should be clean enough, that is so reduced in value by this first operation, that it can be thrown away at once. The cleaning operation, performed in the next cell, is then devoted to securing a high-grade concentrate, with a medium middling, the middling being retreated and sent to the head of the box an indefinite number of times, so that the final product is clean tailing and clean concentrate with no middling at all.

The Janney Machine.—The Janney flotation machine was invented by T. A. Janney, mill superintendent for the Utah Copper Co. The original machine was the mechanical device, as shown in the accompanying illustration. The whole machine is built to a main casting, which sets on either concrete or timber in the middle of the machine. Inside this main casting is a liner casting which takes the wear. This liner is baffled on the lower portion so that the pulp receives violent agitation. Above the main casting is a top liner and support for the motor. A shaft, on which there are two impellers, screws on to the lower end of the motor shaft. This shaft turns with the revolving part of the motor at a speed of about 570 r.p.m. The impellers on the shaft are of two sizes, the lower one, smaller, revolving inside the baffles on the liner creating violent agitation, while the upper revolves just over the baffles and throws the pulp up and out into the spitzkasten, on each side of the agitating chamber.

The Janney machines are of the double spitzkasten type, that is, each agitator serves two spitzkastens. On each side of the agitating chamber, there are two circulating pipes which lead from the bottom of the spitzkasten up and into the agitating chamber. Each spitzkasten is divided by a baffle or partition,

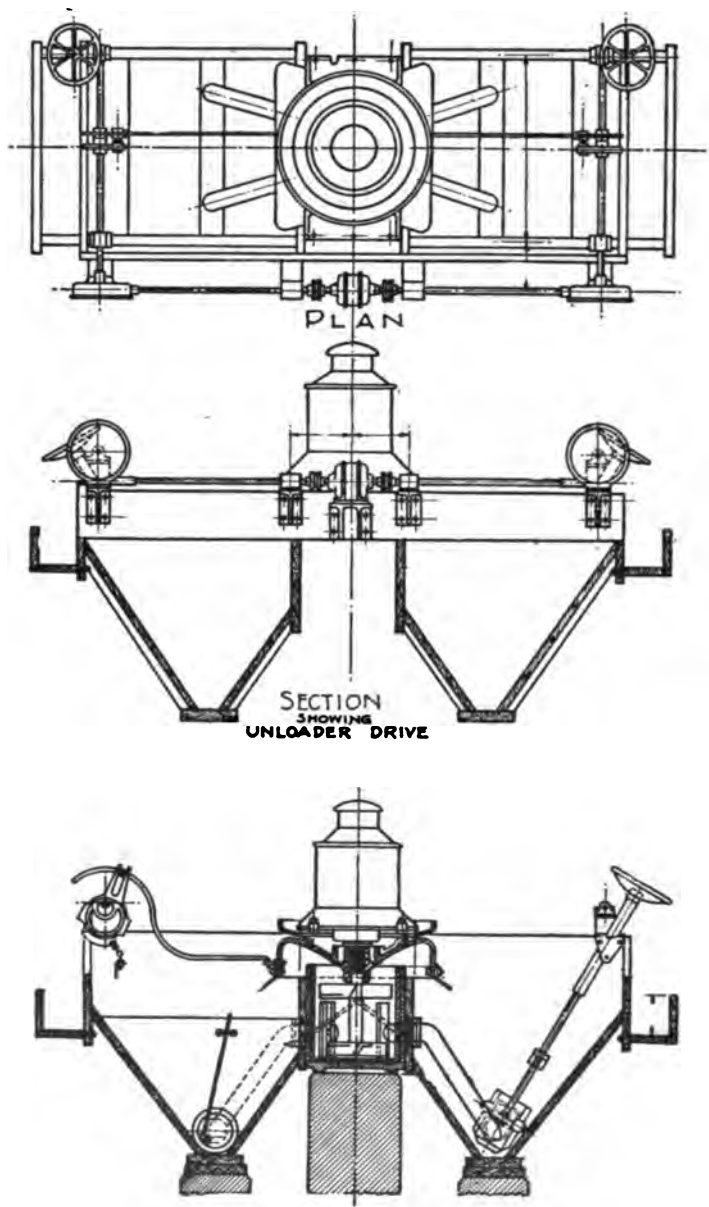


FIG. 19.—Section and plan of Janney "Mechanical" flotation machine.

which prevents the pulp from flowing through the series of machines without first passing into the agitating chambers. A small gate is provided in this middle partition, so that in case it is necessary to shut one machine down for repairs, it may be done without shutting down the whole section. Between the spitzkastens, there is a gate, regulated by a handwheel. By this any height of pulp level may be maintained. When the pulp goes through the gate into the spitzkasten, it passes up through the first circulating pipe into the agitating chamber, where it is

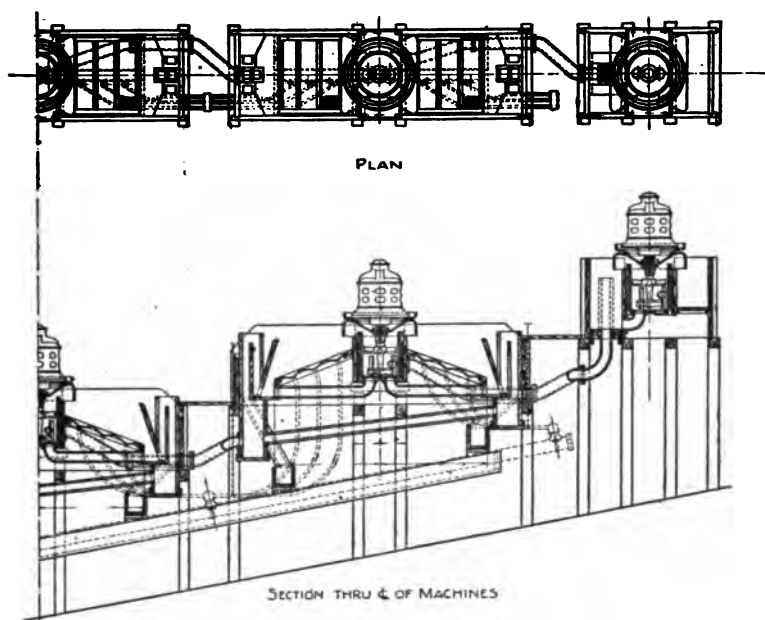


FIG. 20.—Details of Janney "Mechanical Air" flotation machine and Janney Emulsifier.

thoroughly mixed and thrown out into the spitzkasten again. Approximately half of the pulp is thrown back into the upper side of the spitzkasten, and again passes up into the same circulating pipe. Part of the pulp, thrown out into the lower half of the spitzkasten, passes through the second circulating pipe and into the agitating chamber again, and part is advanced through the gate into the next spitzkasten. The rate of flow from one to another spitzkasten depends entirely upon the volume of feed entering the machine. The amount of pulp circulated in the machine fills up to capacity and then remains constant.

feed passes down through a pipe into the agitating chamber of the first flotation cell. Here it receives more agitation and is then thrown out into the spitzkasten on each side. The air being forced up through the air mat, which constitutes the bottom of the spitzkasten, the pulp is again aerated and a light froth produced, which discharges over a lip in front. The sands travel

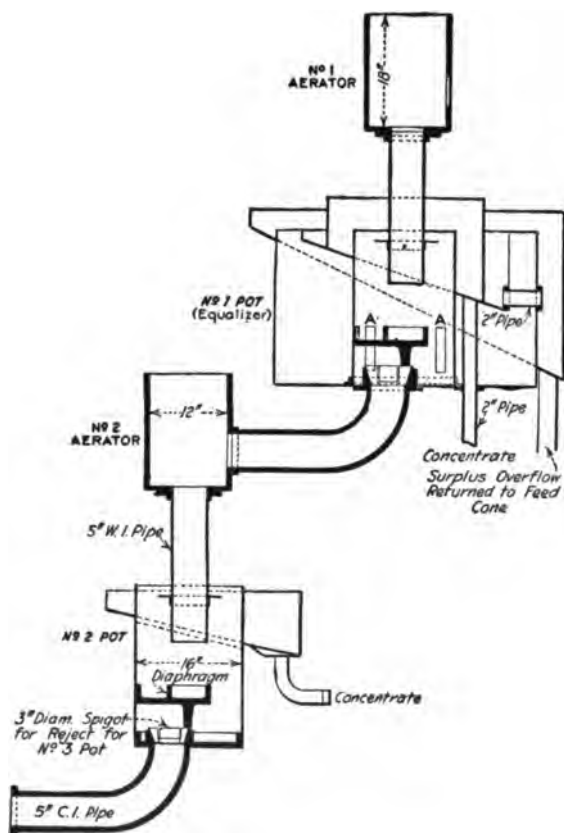


FIG. 22.—Hydraulic-flotation machine used in Australia.

down over the air mat into a box-like arrangement, from which a circulating pipe leads back to the mixing chamber. The tailings discharge from this box over a weir-like arrangement. By regulation of this weir, any depth of froth may be maintained. The accompanying illustration shows the mechanical-air type of Janney machine.

Hydraulic Flotation Machines.—The accompanying illustration, Fig. 21, shows a type of hydraulic flotation machine that was described by Arthur Crowfoot, in the *Engineering and Mining Journal*, Apr. 14, 1917. This machine makes use of no power but uses gravity fall of the pulp for entraining air and producing froth.

Another type of cascade machine, shown in the accompanying drawing, Fig. 22, was used at the Junction North mine, Broken Hill, New South Wales, Australia. It was described in the

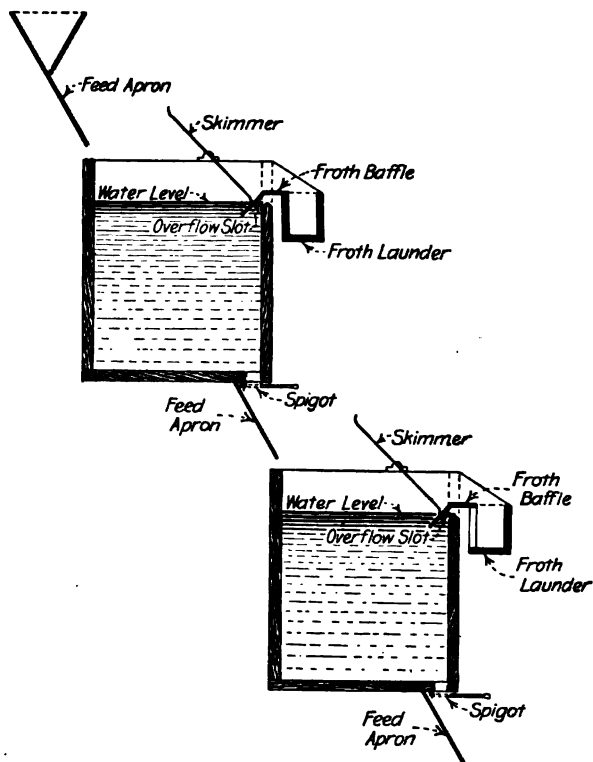


FIG. 23.—C. R. Wilfley's hydraulic-flotation arrangement.

Mining and Engineering Review, of Melbourne, Australia, by T. H. Palmer.

A third type of cascade machine was described by Clifford R. Wilfley, in the *Engineering and Mining Journal*, May 19, 1917. It was developed by Mr. Wilfley at the Mineral Farm mill, Ouray, Colo., in 1916. The machine is said to work out satisfactorily in each case.

The Rork-Sandberg machine, similar to the Kohlberg and Kraut device is a pneumatic device which does not require an extraneous blower or compressor plant and is said to have the advantages of the Janney type of pneumatic agitators. It is the prototype of all machines with a horizontal shaft carrying a partly immersed drum with fan blades distributed around the periphery.

The machine, as described by G. C. Westley,¹ consists of a wooden casing, or chamber, fitted inside with baffles and diversion plates, a form of spitzkasten, a steel shaft with cast-iron spiders carrying fan blades which are keyed or bolted to the periphery of the spiders to form a drum, and means for regulating the flow of pulp from the spitzkasten to the drum section of the cham-

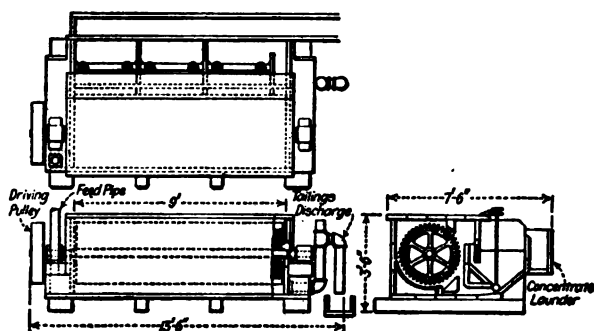


FIG. 24.—The Rork-Sandberg machine.

ber. The machine is constructed so that the shaft is sustained and rotates in bearings supported outside of the casing. A variable-speed motor belted to a pulley attached to the drum shaft operates the mechanism. Ordinarily, in practice the drum rotates between 180 and 200 r.p.m. A feed pipe at the pulley end of the machine receives the pulp charge, and a siphon at the other end discharges the tailings to waste.

In operation the drum, rotating in the direction indicated by the arrow in the accompanying sketch, beats up the pulp as it emerges into the chamber from the feed pipe and progresses in its spiral course to the discharge. At the same time air is drawn into the apparatus from the ends through annular openings about the shaft. The air is forced by the fan-like action of the blades of the drum into the pulp through slits between the fan blades.

¹ *Eng. and Min. Journal*, Feb. 24, 1917.

This action results in the formation of sulphide carrying froth from the emulsified pulp. The mixed froth and pulp is forced over the lip of a vertical partition into the spitzkasten section of the apparatus.

The froth discharges over the lip of the spitzkasten into the concentrate launder, while the residual pulp falls to the bottom of the section and passes back into the drum section of the apparatus for retreatment or cleaning.

If the oil is charged into the pulp mixture passing to the tube mill or other form of pulverizer, no other preliminary treatment is necessary, and the undersize discharge of the grinder may be passed directly into the feed pipe of the Rork-Sandberg machine.

It is obvious that a machine unit of this kind combines the separate functions of a rougher and cleaner. It is plain also that the cell might be made of any length called for by any special kind of work required of it and made adaptable to the demands of differential flotation. The larger unit, whose dimensions are shown in the sketch, is designed to treat between 35 and 40 tons of 80-mesh material per 24 hours. The power required to operate would be about 5 hp.

The fan blades are made of hardwood and experience has shown that the machine may operate continuously eight or nine months before replacement of the worn blades is necessary. Repair is made easy by the provision of a hinged top to the casing, which gives convenient access to all parts of the interior of the casing.

A new machine acting on the same principle has lately been developed by Kohlberg and Kraut, as an improvement upon their original design. This machine is apparently simpler mechanically, easier to manipulate, and capable of more convenient management generally. It is said to have proved quite satisfactory in its operation under plant conditions over considerable periods of time.

The Murex Magnetic Paint Process.—The Murex Process, while not strictly a flotation system, depends upon the selective adhesion of oil to mineral particles, and therefore may properly be considered in the study of flotation processes. The Murex process depends upon the fact, as has been said, that oil adheres to mineral particles, but in this case the oil is highly charged with magnetic iron oxide, which is thus painted upon the mineral sulphide particles. Finely ground magnetic material such

as filings of iron or ground magnetite or pyrrhotite, is mixed into the oil before it is added to the ore pulp, and when the oil adheres only to the sulphide minerals, the effect is that they receive a coat of magnetic paint. By using a magnetic separator adapted to slime separation the particles of sulphide are lifted out of the pulp away from the non-magnetic gangue. Material of sizes hardly regarded as slimes, such as particles two or three millimeters in diameter, can also be treated in this way.

In preparing the oxide or magnetic mineral, it should first be ground to an inpalpable powder, just as paints are made. In this way it is assured that a minimum quantity of the material will cover a maximum area of mineral sulphide particles.

The process has been tested out in the Bergwerks-Wolfart,¹ Clausthal, in the Harz Mountains of Germany, as well as at the Whimwell mine in Australia. Fairly satisfactory results have been obtained, though occasionally baffling difficulties were encountered. The magnetic paint, which is made up of oil-gas tar mixed with twice its weight of finely ground cast iron (any animal, mineral, or vegetable oil can be used, or even a fatty acid, soap, or other such material) is applied in a horizontal rotating cylinder resembling a tube mill except that it is lined with blocks of wood set end on end. The mill contains iron or lead shot and pebbles and the discharge is preferably at the periphery of the end. The metal balls, which are about $\frac{1}{4}$ in. in diameter, become coated with the paint which they in turn rub on to the surface of the ore particles. The function of the pebbles is to prevent formation of granules of oiled minerals or any other lumps or agglomerations, and to prevent the shot from sticking together. It is best to give the pebbles a coating of paint before they are put into the mill. The shot are inspected from time to time, and it has been found that so long as they are well coated the process works satisfactorily. In case the ore requires grinding, the operation can be carried on in an ordinary tube mill.

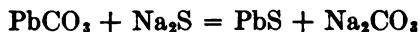
Some difficulty was encountered in the tendency of ground magnetite or other material to leave the oil and go into the water. One of the patents, however, covers the method of preventing this loss of magnetite by using certain additional agents with the oil when the magnetic paint is made up. When using saponifiable oils, all that is necessary is the use of any sulphate, chloride,

¹ Oliver C. Ralston, "Control of Ore Slimes," *Eng. and Min. Journal*, April 29, 1916.

sulphide, hydrate, or other salt soluble in the oil or which forms insoluble soaps with the oil. In case the oil used is a mineral oil that will not saponify, it will require, in addition to the salt, a small amount of soap or saponifiable oil. A small amount of a solution containing 1 to 5% of alum, mixed with the oil, has been found to do good work.

With some ores, it is found that an acid solution works best, and with other an alkaline solution. As a mineral froth does not have to be formed, this process can work through a wider range of solutions than can the flotation process. The plant at Wolfart used a neutral solution and treated a galena ore containing 10% lead, while the Whimwell plant used an alkaline solution and treated a copper ore containing sulphides and carbonates.

The Theory of the Murex Process.—The underlying theory of the process is undoubtedly much the same as in the flotation process, wherein certain minerals are coated with oils while others remain uncoated. In fact it has not been proved that the Murex process will treat any ore which the flotation process cannot treat, unless we accept Hyde's statement that under proper conditions it is possible to treat lead carbonate by the process. In this connection Hyde probably had in mind U. S. patent No. 996491, taken out by Lockwood and assigned to the Murex company, in which is given a method of applying the Murex process to the concentration of oxides and carbonates. This consists of the use of a soluble alkaline sulphide in solution to form a film of metallic sulphide over the surface of the carbonate particle by reaction with it. Thus the reaction with lead carbonate of an ore is as follows:



Lockwood gives two examples of the method of working. A hundred pound lot of ore containing carbonate, phosphate, and chloride of lead was crushed to pass at 24-mesh screen and then suspended in 80 lb. of water. To this was added 4 lb. of finely ground magnetite in 2 lb. of Texas residuum oil. After agitation for about 10 minutes, a solution containing about 3 oz. potassium sulphide was added. A film of lead sulphide was formed on the oxidized lead minerals and they were successfully coated with magnetic paint ready for separation. The

second example is of the sulphide filming of a carbonate ore of copper—likewise successful.

As the sulphide filming of carbonate ores of lead and copper is now meeting with considerable success in the flotation process, it may be said that the Murex process has a limited field. Lockwood has presented a method of treating slimes of mixed materials in order to avoid having to grind them further. Middlings of galena and sphalerite can be treated with a solution of caustic alkali and silicate of soda with the result that the galena and sphalerite separate along cleavage lines without the need of further crushing. Supposedly this is due to the tendency of sodium hydrate and silicate to disperse a suspension of negatively charged particles. The increase in degree of dispersion of clays by the use of these chemicals is fairly well known. After feeding the galena and sphalerite in this way, they can be separated on a slime table or on a vanner. He proposes the same method in tabling of mixed zinc lead flotation concentrates.

Lockwood gives, as an example of this method applied in the treatment of ore, two tests that were run on two tons of ore from the Broken Hill Proprietary. The ore was crushed to about 40 mesh and divided into two lots. One lot of 2,240 lb. was treated with 500 lb. of a solution containing 5% of caustic soda and 1% of sodium silicate. After agitating for 30 minutes, the pulp was fed onto shaking screens, and the material remaining on the 60- and 80-mesh screens was run over vanners. The results are shown in the first table. With the other lot of 2,240 lb., the same conditions were observed except that pure water was used on the ore. The results are tabulated for comparison.

SEPARATION BY USE OF SODIUM HYDROXIDE

Wt., lb.	Size	Concentrates		Middlings wt., lb.
		% Pb	% Zn	
103.5	On 60	76.8	3.9	43.5
172.0	On 80			36.5

COMPARISON BY USING PURE WATER

Wt., lb.	Size	Concentrates		Middlings wt., lb.
		% Pb	% Zn	
88	On 60	72	6.5	130
134	On 80			68

Lockwood examined the middlings in each instance and found that in the case of the dispersed pulp, they seemed to be made up

of free crystals of galena and sphalerite, while the middlings from the test in water seemed to be true middlings, containing composite particles of the two sulphides that required further grinding. It is quite possible that floccules containing both sulphides could be dispersed by the sodium salts and that it was hence easier to get a good clean tabling test on this account. The idea certainly merits further investigation.

CHAPTER VI

TESTING ORES FOR FLOTATION

Like all other details pertaining to flotation, the procedure of testing an ore to determine its availability for treatment by the process has not been reduced to a definite series of operations. There are a multitude of ways in which the tests may be made, and it seems probable that satisfactory results can be obtained by any of them, provided they are carried out with judgment and with a view to the peculiar requirements of the ore under investigation.

In treating of the subject of testing ores by flotation, Hoover¹ advises that for one who has never undertaken such tests the best results would be obtained by starting with a very simple ore, upon which the process is clearly and easily applicable. He mentions a mixture composed of about 60% quartz and 40% hornblende, or 70% quartz and 30% of pyrite or chalcopyrite. He calls attention to the fact that these mixtures are not common in nature, but can be closely approximated by some well-known ores. This seems to be a fairly good way to start out since it will enable the operator to become acquainted with the action of ores under flotation tests; with the appearance and character of froths, and with the mechanical manipulations necessary to secure satisfactory results from the work. Of course, in carrying out tests of this sort, some equipment is required, but very good work can be done without any serious outlay for machinery. It seems, then, a reasonable method of explanation to begin with the machines and appliances that may be used for testing ores by the flotation process.

The Requirements for Flotation Tests.—Since the requirement for effecting flotation is a combination of oils, finely ground ore, water (and perhaps acid or some other agent in the solution), and a method of thoroughly mixing the whole, it may be seen that they are rather easily fulfilled. Successful tests can be made by shaking up the combination in a bottle or in a Mason

¹ "Concentrating Ores by Flotation," page 62.

fruit jar, but these appliances have the objection that the separation of concentrates from tailings is not easy.

As a first step in progression beyond the period of shaking a bottle, the separating funnel, a rather common piece of laboratory apparatus, has been successfully used for the purpose. Its use is described by Ralston and Allen,¹ who show that by giving a thorough shaking in a separatory funnel the froth can be formed, and the appliance has the additional advantage that the tailings and water can be drained off through the stem of the funnel, leaving the froth and concentrates in it. A sharp line of demarcation can be taken advantage of with this appliance. It has the additional advantage that, being made of glass, the operation is visible at all stages, and judgment can be used as to when agitation has been carried to a sufficient point.

The Slide Machine.—Perhaps next device in order of importance is the slide machine that was invented by Hoover and perfected by a series of others since his time. As described by Hoover in his book already referred to, this machine is made in two parts, an upper one *A*, and a lower one *C*, both sliding easily on the rubber cushion *B* between the two parts, or, better still, having the bearing surfaces highly polished. The rectangular section of the interior is $4\frac{1}{2}$ in. The upper part *A* has a tail *D*, the purpose of this being to prevent leakage of the froth when the upper half is slid to the right for the purpose of removing it. Windows of glass in each of the two parts enable the operator to watch the progress of the test. At *E* there is a simple agitator with four arms at right angles. The shaft driving this agitator goes to the bottom *G* of the lower part *C*, and is tapped with a small stuffing gland to prevent leakage. At *F* is a hole communicating with the interior at the bottom, closed with a valve. This hole is for the withdrawal of tailings. The agitator arms are 2 in. long from the center of the shaft, thus describing a circle of 4 in. when revolving, and so leaving a clear space of $\frac{1}{4}$ in. between the ends of the arms and the sides of the interior. The normal speed of this agitator is about 1,500 r.p.m. The machine should be made of some metal that gives a clean casting. The lower surface of *A*, which rests on the rubber cushion *B*, and the upper surface of *C*, upon which the rubber cushion rests, should be planed and polished smooth. The

¹ "Testing Ores for the Flotation Process," *Min. and Sci. Press*, July 24, 1915.

cushion is a piece of rubber insertion about $\frac{1}{16}$ in. thick, and just wide enough to fill the space between the edges *H* and *J*, of the tray-like part *C*. The cushion can be discarded if the surfaces are accurately planed and polished. The accompanying illustration shows the construction of this machine, and its operation, which is simple enough, can be readily understood. After forming the froth by agitation caused by high-speed revolution

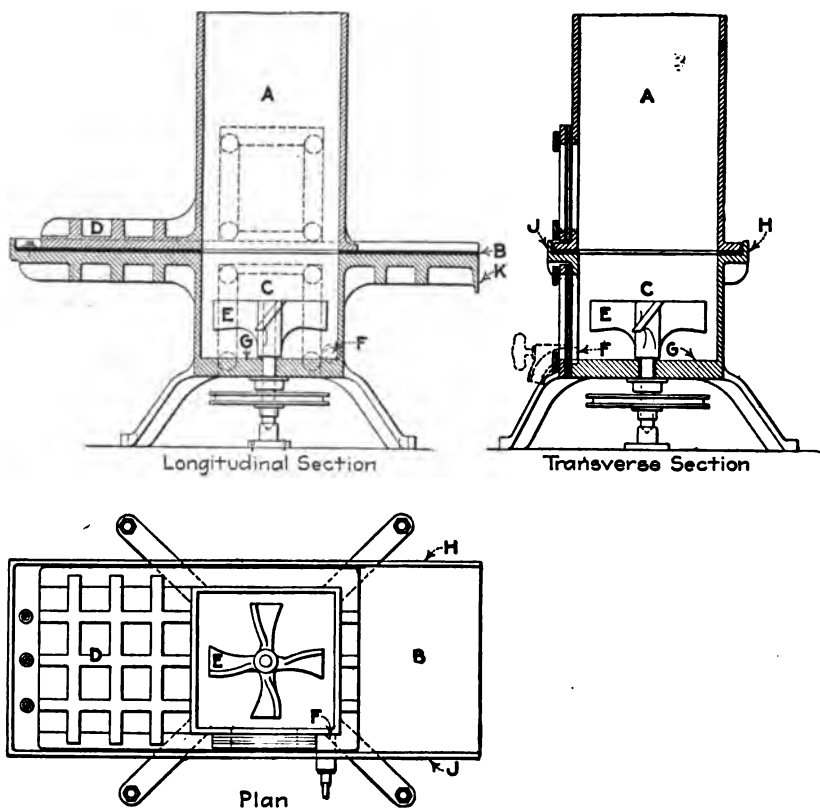


FIG. 25.—The Hoover slide machine.

of the paddle, the froth is brought above the dividing line where the machine separates. Having placed the line of demarcation between foam and pulp to agree with the division of the machine, the top half is slid off to one side, carrying the froth and concentrates along with it, the tailings remaining in the lower half, from which they may be drained off through the bottom.

Other Experimental Machines.—In addition to the machines already described, miniature copies of the Minerals Separation machine may be obtained for testing purposes, and likewise small models of the Janney, the Wood, the Potter-Delprat, Callow, and nearly all the other devices. The unit in the commercial Macquisten machine is so small that a single tube will serve very well as an experimental installation. Besides these

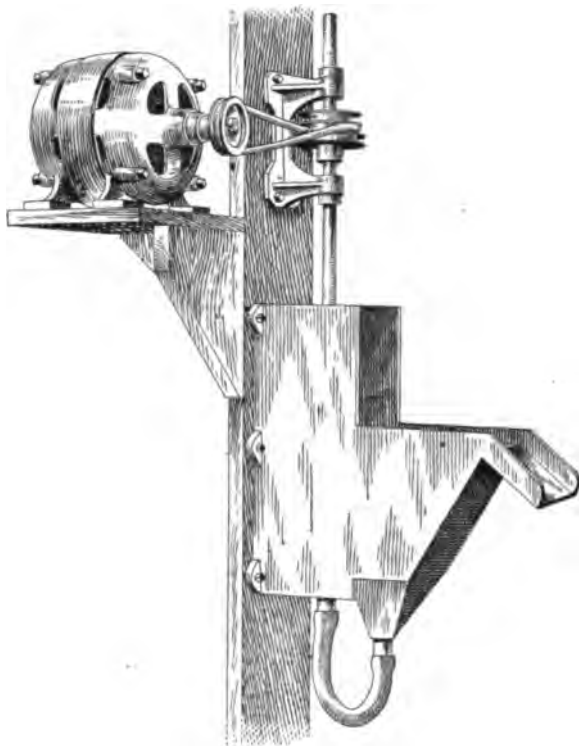


FIG. 26.—Case testing machine.

machines there are several made especially for experimental purposes, such as the Case, and that of Roy & Titcomb of Nogales, Ariz. Many experimenters, however, prefer to build their own flotation machines, and this is a simple matter. It may readily be done by using one of the well-known types as a model and building the machine of wood. The principal objection to the home-made machines is that they usually do not contain glass

windows in the sides, although by the utilization of skilful workmanship such conveniences may be incorporated in them.

Flotation Factors of an Ore.—The examination of an ore for its applicability to flotation treatment will involve the study of about six points. In the order of their importance these are as follows: (1) the size to which the ore must be crushed; (2) the dilution ratio; (3) the temperature at which the solution must be carried; (4) the amount of acid required; (5) the question of oil, this being divided into two parts, the first of which deals

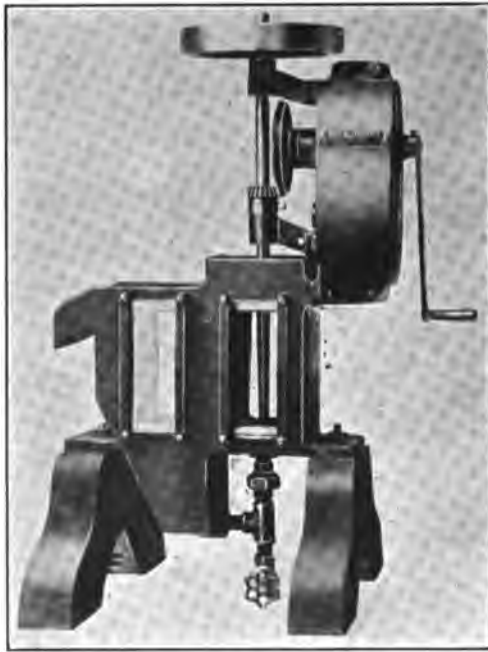


FIG. 27.—Roy & Titcomb testing machine.

with the variety or kind of oil or oils to be used, and the second dealing with the amount required; (6) the agitation required, its kind, duration, and comparative violence.

Before attacking any of these details directly connected with flotation, of course, it must be ascertained that the sample upon which experiments are to be carried out is a representative one of the ore to be treated. This applies as well to flotation as to any other metallurgical process, and has been so thoroughly insisted upon by metallurgists that there would be no good purpose

served in going further into it here. It is sufficient to say that it is essential to have a representative sample, since without it the metallurgical results, no matter how carefully obtained, are altogether worthless.

In addition to the accuracy of the sample, its condition is also a matter for careful consideration. If material for flotation is in the form of a pulp already ground by milling machinery, then it is essential that flotation tests be carried out on the pulp as it

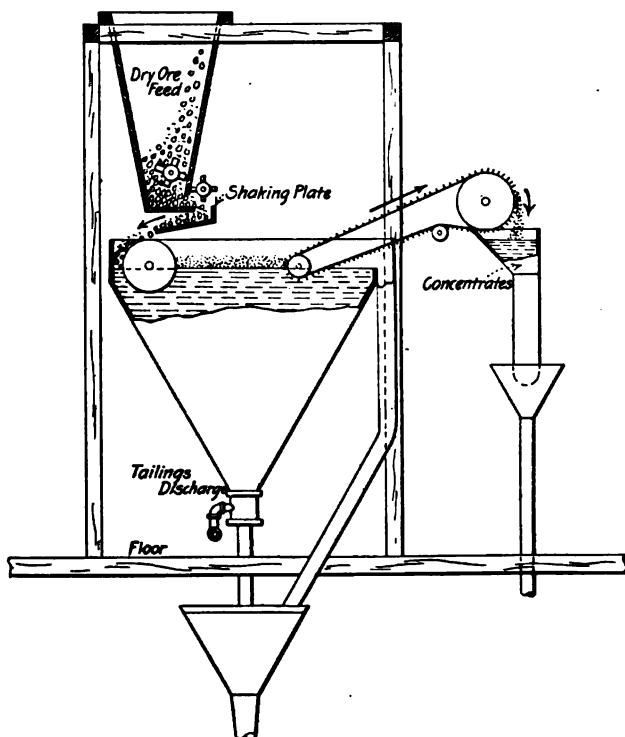


FIG. 28.—A testing machine of the Wood type.

occurs. Should the solids be separated from the mill fluids, a change and opportunities for error are introduced. Should the solids be shipped to a distant testing laboratory an error is almost sure to occur, since even superficial drying may change the surface conditions of the particles so as to increase or decrease their tendency to float, thereby vitiating the result of any experimental work.

Tests Preliminary to Flotation.—Before attempting flotation, it is also well to carry out a further series by a preliminary series of experiments to determine whether or not part of the ore can be treated by flotation and a part more economically by some other process. This, of course, depends upon the kind of ore under consideration, and with some particular classes the question may be passed over without detailed examination. On many ores, however, it is well to carry out a series of grinding tests to find out whether or not a certain portion of the ore, whether it be the coarse or fine portion or intermediate material, can be discarded without further treatment or can be handled by a treatment which is cheaper than the one under consideration. For example, on many concentrating ores it is found that the portion above a certain mesh—take 40 mesh for example, merely as an illustration—can be classified as to size and concentrated on tables to the best advantage, leaving a residue that would be too low in value to be treated by any other means. In this case, only the portion under 40 mesh need be handled by flotation, and this at once cuts out a large portion of the tonnage, makes treatment costs lower, and appreciably reduces installation expense.

In addition there is a series of experiments that ought to be carried out on the slimes portion. A study of the series of articles on "Colloids in Ore Dressing"¹ shows that a differential system of flocculation and settling may possibly be carried out in such a way that perhaps the greater part, and sometimes practically all, of the valuable constituents can be segregated into a definite portion of the slimes. This will reduce the tonnage necessary to be treated by flotation and will naturally reduce installation expense as well as treatment cost. This question should be carefully investigated, since it is by no means impossible to carry out preliminary experiments toward a conclusion that will lead to a material simplification of the actual treatment.

Actual Flotation Experiments.—In considering, however, the actual treatment by flotation itself, the points that have been enumerated will be the ones upon which a final decision will have to be based. In carrying out of such experiments, Hoover recommends a series, the first experiment of which is the crushing of ore to 60 mesh and testing in the slide machine at a ratio of dilution

¹ *Eng. and Min. Journal*, Feb. 5, 1916, and subsequent articles.

of 1:3, using a small quantity of acid and a frothing oil. This experiment is repeated with the same kind of ore at the same dilution but with about ten different variations of the oil, the recommendation being for the primary use of oleic acid. In each case the concentrate should be removed, weighed, and assayed. It is well known in flotation processes that a high-grade concentrate and high percentage of concentration can seldom be procured at the same time. For this reason the usual practice is to first secure the highest possible percentage of concentration, removing from the ore the greatest possible quantity of the valuable minerals, and thereafter retreating the low-grade concentrate formed in order to eliminate as much as possible of the gangue and worthless material carried. This system is usually referred to as roughing, which is the first operation, and cleaning, which is the succeeding one. Cleaning, however, may be done any number of times, and in actual practice is often done three or four times and sometimes more. It is often possible, also, after having produced the first or rougher concentrate, to effect a selective flotation of the concentrates, by this means separating the constituents when two or more valuable minerals are to be saved. This is found particularly true in cases where zinc, lead, and perhaps some other minerals occur in combination, a good example of such ores being found in the Coeur d'Alene district of Northern Idaho.

In carrying out experiments one soon becomes accustomed to the behavior of froth and can tell at a glance whether or not good results are being obtained. By this means it is possible to save considerable time by eliminating the experiments that are plainly not producing good results and this procedure will materially lessen the expense attached to testing.

Varying the Oil Factor.—After carrying out the first experiment as has been noted, another experiment should be started maintaining all the factors exactly the same except for the kind of oil, and in this way the best oil applicable to the ore is found at first. Naturally, the selection of the oil is not the first in logical order of sequence, that place belonging to the stage of grinding required to secure good extraction of the ore. It is possible, however, to secure fairly good extraction from most ores that are ground to 60 mesh, and by determining definitely the best oil or oil combination to be used, or that which produces the best results, taking everything into consideration, at a grind-

ing to 60 mesh, that experiment is pretty sure to provide an important basis for subsequent experiments. An oil that will produce the best results with the ore ground to 60 mesh will probably produce the best results with the same ore no matter to what mesh it is ground. That is not to say that the extraction percentage will be the same, because it naturally will vary with the grinding, but the fact remains that the oil best applicable to a certain ore at about 60 mesh will very likely be applicable and give best results no matter to what mesh the ore is ground. Having determined the kind, or combination, and quantity of oil to be used, this factor may be kept constant in all subsequent experiments. It is a good plan, however, after having fixed all other factors to go back to the oil again and determine whether or not any further variation will give better results.

In carrying out flotation experiments, as with any others, it is best to keep complete records and very full notes, even describing in the record book the appearance and physical action of the oils, pulp, froth, and every other detail that attracts attention.

Having determined the quantity and kind of oil to be used, the question of acid can be taken up, since that also is one that will not vary with the fineness of the ore. The use of acid is a factor that can almost always be determined roughly by forecasting, but experiments will have to be carried out to determine the quantity required. In a great many cases no acid at all is used and in other cases it is found to be required. The determination of this factor, however, is not difficult and may be carried out in a very short time. The same is true also with the dilution of the pulp. Usually it makes very little difference what the dilution may be, but it is important that that dilution should be constant throughout all work.

Effect of Grinding on the Tests.—After having determined the oil, acid, and dilution best adapted to the flotation treatment of the ore under consideration, the next point to be determined is the degree of grinding that produces best results. In carrying out experiments of this kind, it is considerably better to take an ore that has been ground in practice, rather than one that is ground in the laboratory. Laboratory grinding gives a product that does not duplicate the work of stamps, rolls, tube mills, or whatever grinding machine is used in practice in the mill. If, however, laboratory grinding must be resorted to, the best way

to perform it is to crush the ore in a great number of very short stages. To carry out work of this kind, a coarse ore should be put on the bucking board and broken down either with a muller or with a hammer until most of it is reduced to about $\frac{1}{4}$ -in. size. The whole should then be screened, removing the product that has already reached the point to which grinding is to be carried in that particular experiment. If the first experiment is to be on 20-mesh product, a 20-mesh screen should be used at once to take out the rock that has already been ground that fine. Another crushing stage is then carried out, using breaking as far as possible instead of grinding, or rubbing, which may produce erroneous results in the subsequent experiments. The rubbing or grinding process is likely to change the angular state of the rock or mineral, a point that is of extreme importance in the flotation process. Ores that are ground in practice through rolls, stamps, tube mills, etc., are not subjected to this rubbing action and produce material that maintains its angular character to very high degree. Bucking the material on a bucking board has a tendency to round off the mineral grains, changing their shape and consequently their action under flotation conditions. This is a point that should not be overlooked, but which up to the present time has not been given a great deal of attention. Frequent screening should be carried out in order to avoid deformation or overgrinding of the rock particles. Of course the final grindings will probably have to be bucked, or rubbed, but while this does introduce a small factor of error, it is one that cannot be avoided and all one can do is to take the best possible care to have as little of that sort of manipulation as possible.

A series of tests on different screens varying between 20 and 100 and depending upon the ore to be treated and the conditions in general, will probably locate the best possible degree of subdivision. Included in the grinding experiments should also be separation tests such as have already been mentioned, setting off the slimes to one side and the sands to another and determining whether or not treatment of different kinds could profitably be applied to the two products. In the whole field of ore testing, there is probably nothing so enlightening as screen classification, assay of the products and experimentation upon them by different metallurgical processes. It is a procedure upon which not nearly enough attention has been concentrated, but no other

one point of study will throw so much light upon the character of an ore and its action under treatment than a thoroughly carried out and highly developed screen analysis, with metallurgical experiments upon the products.

Temperatures for Flotation Work.—Then, having determined the point to which grinding should be carried, another series of experiments should be carried on to find out the best temperature for flotation. In many cases heat is required, and in some cases to a high degree. In most instances, however, it will be found that by giving careful attention to details, maintaining dilution, and in general taking good care of the operation of the process, normal temperatures can be used without bad results. Following that, the series showing the amount of agitation to be used should be carried out, and in all of these series of tests, a wide range of variation should be attempted, thus securing as preliminary information a very thorough and valuable idea of what can be obtained in practical working.

In testing ores for flotation, it should be carefully borne in mind that experimental results, secured in the laboratory, will seldom or never reach the degree of perfection that can be obtained by large-scale mill practice. The flotation process is in this way a good deal like cyaniding and different from most other processes, by means of which careful work in the laboratory will secure better results than can usually be obtained in the plant. Both flotation and cyaniding, however, give their best results in large-scale work in the plant.

Examples of Testing Practice.—In order to illustrate the carrying out of testing experiments, a good example may be quoted from the publication by H. J. French.¹

In these tests the slide machine, already described, was used and a thorough description of the method followed was outlined. The machine was filled with water to a level slightly above the middle. If heat was required, it was obtained by repeatedly pouring boiling water into the machine until the walls became hot, and then it was filled with water near the desired temperature, which could thus be maintained. The ore was then agitated for a few seconds until thoroughly wetted. When the mixture reached the desired temperature, further heat being obtained from Bunsen burners if required, acid was added in

¹ "Flotation Test on Ores from Bisbee and Cobalt," *Columbia School of Mines Quarterly*, Nov., 1914.

the proper proportions, then the oil was added and the motor turned on, bringing it gradually but quickly to full speed. After the required agitation had been given, the agitator was stopped and the mixture allowed to settle for about a minute, thus allowing time for the froth to collect upon the surface of the water and for the gangue to settle. The upper part of the machine was then slid along the plane of contact, carrying with it the froth and part of the clean water above the plane of junction of the two parts. The process could be repeated on the tailings without the addition of more oil, but before further agitation, water should be added to just above the sliding plate. The tailings and remaining water were then withdrawn into a pan through the bottom spigot, allowed to settle, and the greater part of the water decanted. The tailings were dried, screened, mixed, sampled, and finally assayed or analyzed. The experiments were carried out about as has been noted in this description.

Experiments on the Bisbee Ore.—The Bisbee ore under treatment contains about 65.1% insoluble; 10.9% Fe; 0.9% CaO; 0.28% Al_2O_3 ; 6.04% Cu, and 12.7% S. After testing the ores in the slide machine, the tailings were assayed for sulphur, whence the percentage of extraction of the sulphides was computed. This is a very simple way of getting at it, and efficient in the case of these copper-sulphide ores. The accompanying tables show clearly the effect of the variation of the different items, and show what may be expected in experimenting upon an ore of the kind.

TABLE 1.—EFFECT OF VARIATION IN AMOUNT OF OIL

Pine-tar oil. 100 grams ore. 3-min. agitation. Room temperature. 35 drops H_2SO_4 . 1500 c.c. water. Analyses on 1-gram sample.

Wt. of oil, grams	Wt. of froth, grams	Wt. of tails, grams	Wt. of BaSO_4 , grams	Wt. of S in tails from 100 grams ore	% S in tails	% S extracted
0.5	3.5	97	0.8624	11.85	93.2	6.8
1.0	4.0	97	0.8610	11.81	93.0	7.0
1.3	4.5	97	0.7480	10.27	80.8	19.2
2.0	10.0	92	0.7290	10.00	78.7	21.3
2.5	11.5	91	0.6885	9.46	74.5	25.5
3.1	12.0	90	0.6308	8.66	68.3	31.7
4.0	13.0	91	0.6182	8.50	67.0	33.0
6.0	20.0	86	0.5329	7.30	57.5	42.5

TABLE 2.—EFFECT OF VARIATION OF AMOUNT OF ACID

Pine-tar oil, 3 grams. Ore, 50 grams. Agitation, 3 min. Room temperature. Analyses on 1-gram samples.

Drops of H_2SO_4 in 1500 c.c.	Wt. of tails, grams	Wt. of $BaSO_4$, grams	Wt. of S in tails from 100 grams of ore	% S in tails	% S extracted
None	39.5	0.7320	10.10	80.0	20.0
50	29.9	0.5231	7.17	56.5	43.5
75	41.0	0.4927	6.75	53.1	46.9
100	38.0	0.4147	5.68	44.8	55.2
150	39.5	0.3952	5.43	42.8	57.2

TABLE 3.—EFFECT OF VARIATION IN TIME OF AGITATION

Pine-tar oil, 3 grams. Ore, 50 grams. Room temperature. 100 drops acid in 1500 c.c. of water (= 0.3 %). Analyses on 1-gram samples.

Time of agitation, min.	Wt. of froth, grams	Wt. of tails, grams	Wt. of $BaSO_4$, grams	Wt. of S in tails from 100 grams ore	% S in tails	% S extracted
3	19	37.0				
6	8	48.0	0.620	8.50	67.0	33.0
10	20	36.0	0.404	5.53	43.5	56.5
15	16	39.0	0.804	11.00	86.0	14.0
20	12	44.0	0.503	6.90	54.5	45.5

TABLE 4.—EFFECT OF VARIATION IN TEMPERATURE

Pine-tar oil, 6 grams. Acid, 5 grams. Ore, 50 grams. Agitation in two 5-min. periods, skimming after each period. Analyses on 1-gram samples.

Temp. °C.	Wt. of froth, grams	Wt. of tails, grams	Wt. of $BaSO_4$, grams	Wt. of S in tails from 100 grams ore	% S in tails	% S extracted
25	20	36	0.850	12.0	92.0	8.0
40	29	27	0.535	7.3	57.0	43.0
50	22	34	0.640	8.8	69.0	31.0
65	14	42	0.450	6.2	48.6	57.4

TABLE 5.—SCREEN ANALYSIS OF BISBEE ORE

Screen opening, in.	Mesh approximate	Wt. per 100 grams of ore	Wt. of BaSO ₄ from 1-gram samples	Equivalent sulphur %
0.0116	48	4.0		
0.0082	65	13.5	2.1038	28.90
0.0058	100	21.8	1.2003	17.04
0.0041	150	14.7	1.0022	14.05
0.0029	200	14.0	0.9175	12.60
Below 0.0029	-200	32.0	0.7124	9.80

TABLE 6.—EFFECT OF VARIATION IN SIZE OF ORE

Ore, 50 grams. Acid, 0.3%. Pine-tar oil, 6 grams. Temperature, 65°C.
Two 5-min. agitation periods. Analysis on 1-gram samples.

Mesh	Wt. of froth, grams	Wt. of tails, grams	Wt. of BaSO ₄	Sulphur in tails, %	Sulphur ex- tracted, %
48- 65	12	44	1.1080	57.3	47.7
65-100	28	28	0.3396	27.3	72.7
100-150	26	30	0.1294	10.3	89.7
150-200	19	37	0.3496	38.0	62.0
Below 200	19	37	0.2500	35.0	65.0

TABLE 7.—EFFECT OF VARIOUS OILS ON COBALT ORE

Ore, 100 grams. Water, 1500 c.c. Agitation, 1 min. Room tempera-
ture. Oil, 8 drops. Acid, 1 gram.

Oil	Wt. of tails, grams	Wt. of froth, grams	Oz. Ag. per ton tails
Pine-tar.....	97.0	3.0	9.60
Fish.....	98.5	3.0	12.46
Sperm.....	98.0	2.0	12.94
Corn.....	84.0	16.0	13.36
Whale.....	98.0	2.0	13.80
Cylinder.....	(a)		
Rapeseed.....	90.0	10.0	10.87
Lard.....	90.0	10.0	13.13
Cottonseed.....	93.0	7.0	12.66

(a) No froth.

TABLE 8.—EFFECT OF VARIATION IN AMOUNT OF OIL

Ore, 100 grams. Water, 1500 c.c. Acid, 1.5 grams. Room temperature. Agitation period, 2 min. Pine-tar oil.

Oil, grams	Wt. of froth, grams	Wt. of tails, grams	Ag in tails, oz. per ton	% Ag remaining in tails	% Ag extracted
0.3	7.0	93.5	11.44	84.0	16.0
0.6	12.0	89.0	10.20	75.0	25.0
1.0	15.0	84.5	8.25	60.6	39.4
1.3	11.5	90.0	8.28	60.8	39.2
1.6	15.0	87.0	8.68	63.8	36.2
2.3	16.0	87.5	6.66	49.0	51.0
3.0	25.0	78.0			
3.3	22.0	81.0	6.45	47.5	52.5

The Cobalt Ores.—A series of tests were also carried out on characteristic ores from Cobalt, the particular samples being from the Coniagas Mining Co. The ore contained about 13 oz. of silver per ton, mainly in the form of pyrrargyrite, proustite, argentite, dyscrasite, and native silver. The procedure was about the same as for the Bisbee ores, but in this case the tailings were analyzed for silver. The accompanying tables show the results obtained in the work upon the Coniagas ores. This work is an example of good procedure and enlightening results acquired through careful experimentation. It must be borne in mind, however, that they were carried out a long time ago, in comparison with the rate of progress of the flotation process. If done at the present time, further investigations would undoubtedly be carried out, since a great deal more has been learned about flotation and its requirements in the last year or two. It is true, nevertheless, that this work is a type of good experimental flotation procedure, and would give a vast amount of information about the action of the ores even at the present time. In fact it might be quite sufficient to show the best that could be done with the ores.

Experimental Flotation Machines.—The slide machine has been very largely used for experimental flotation work for a long time, but, due to the convenience of the more modern experimental machines and to the fact that they produce results more nearly comparable with those obtained in plant practice, it has gone almost altogether out of use. The small machines

designed on the same lines as the large apparatus used in the plant have almost entirely taken its place. Many of these machines are to be had on the open market at the present time, or they may be put together rather inexpensively at the plants where experiments are to be carried out. As has been said, they come nearer duplicating plant results than the slide machine does, since they actually agitate the ore in a practical way to produce the froth, and the froth then overflows into some convenient receptacle, just as it does in practice.

In the operation of any of these machines that are designed to make an oil froth, enough clear water is run into the machine to show in the spitzkasten and then the motor is started. The ore charge is then added and a thorough mixture secured. At this point the machine may be stopped and other ingredients added if required, but if not, water is added until the froth begins to overflow from the lip. It is usually better to secure thorough agitation before the froth is allowed to overflow, and when it does overflow, it may be allowed to do so of its own accord or it may be assisted by skimming with a paddle or some other convenient utensil. As a matter of fact some of the practical machines use skimming devices to assist in the removal of the froth as it is formed. The duration of agitation in the ordinary test is about five minutes but this is a variable quantity and cannot be taken as even approximately fixed, because some ores require considerably more, while others will produce their maximum of froth in a very short time.

The Callow Testing Method.—In carrying out tests for flotation by the Callow pneumatic process, it is best to do so in a machine of the same type. Miniature Callow machines are supplied for this purpose, and may be had of appropriate size to carry out experimental work. They are usually provided with a primary mixer—a Pachuca tank—such as has been used in practice as an integral part of the process. The Pachuca tank is exactly like those used in the cyanide process for agitating slimes and consists of a long, cylindrical cone-bottomed tank having a central air column. Agitation is by introduction of air under low pressure to this central column, making the pulp in the column lighter, on account of which it rises, the mass of pulp in the rest of the tank falling in proportion. The Callow test machine is really no more than a miniature of the regular installation. The air is subdivided into fine particles by a pas-

sage through the canvas filter bottom and passes up through the pulp, raising the oiled sulphides that become attached to the air bubbles. In carrying out tests with this machine, there is nothing that has to be changed, as compared with testing in other devices, except the operation of the machine itself. A clear understanding of the Callow flotation machine as used in mills following the process is all that is required to successfully carry out small-scale experiments with it. The procedure of



FIG. 29.—A Callow testing unit.

testing does not change at all, the only difference being in the machine manipulation.

In carrying out this process the mixture of ore, water, oil, and other reagents is fed into the suction of the sand pump or into the top of the Pachuca tank after air has been started into the various machines. Sometimes the Pachuca tank is omitted, provided proper mixture can be obtained before the pulp is put into the frothing machine. In practice it has been found quite possible to leave the Pachuca tank out of operation, thorough mixing being secured by the pumping and general agitation as the pulp passes through the machinery preceding the flotation

plant. The overflow from the Pachuca tank, when it is used, into the rougher cell accumulates until a good froth comes up and nearly overflows. Then the tailings discharge-valve in the rougher is gradually opened and the froth allowed to overflow from the cell into the cleaner cell. It is considered best practice to get most of the charge circulating before a great deal of the froth is allowed to overflow, the overflow of froth being controlled by the main air valves leading to each unit. Once adjusted, these valves leading to the individual air boxes in the bottom of the machine should never be disturbed, and all control of air supply should be at the valves in the main pipe. When everything has been adjusted to smooth operation, the air pressure in the cleaner can be increased until concentrate froth is overflowing into the vacuum filter, a small miniature one being installed at the tail end of the flotation machine. This filter may be of the simplest kind, even an ordinary glass funnel with filter paper set in the laboratory suction pump will answer the purpose satisfactorily. Any settled material in the flotation cells should be stirred up, which can be done with a small stream of water, or with a glass rod or wooden stick.

All the foregoing description has applied to those flotation processes which use oil, and have froths formed by mechanical or air agitation of the pulp. The other processes—film flotation—that are in practical use include the Macquisten and the Wood processes. These are the principal ones and, although there are some others—notably the Potter-Delprat—actual large-scale practice is practically confined to the Macquisten and Wood processes. The Wood machine can be built in miniature, as is the case with other machines of the kind that have been used to carry out experimental work. The method of operating is the same as that of a full sized machine. The experimental device is about as shown in Fig. 28. As in this process the surface tension is the only factor affecting results, the work has to be carried out with extreme care, just as it does in practical work at the plant.

As to the Macquisten process, a single tube of the ordinary commercial size is usually used, and is plenty small enough to secure results on an experimental scale. No further explanation of that machine is necessary.

The Potter-Delprat Method.—The explanation of testing ores by the Potter-Delprat process is best given by Ralston and

Allen.¹ An illustrative test is described by them. Tubes containing 3% sulphuric acid, or acid sulphate solutions and a little sulphide ore are warmed nearly to boiling temperature. Bubbles of CO_2 attach themselves to the sulphides, travel to the surface of the solution, discharge into the air, and drop the sulphides into the pocket on the underside of the tube, made for the pur-

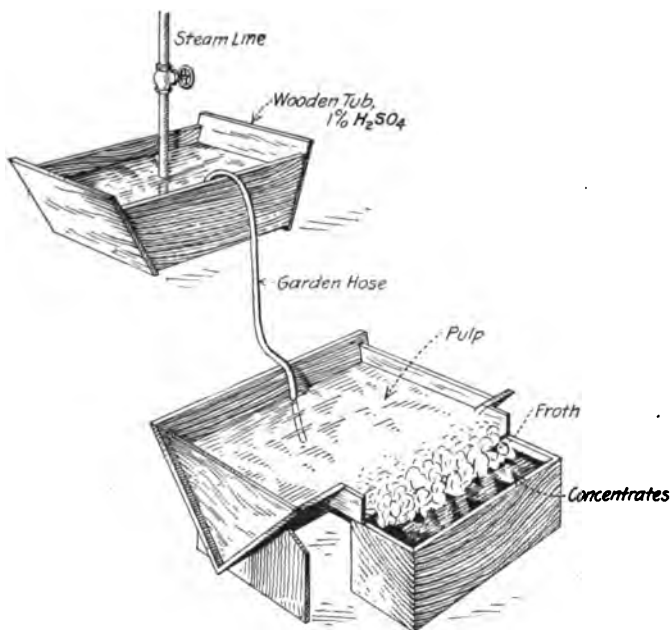


FIG. 30.—A test plant for the Potter-Delprat process.

pose. In another test a 200-c.c. beaker is used with 100 c.c. of 3% sulphuric acid, and brought to nearly boiling temperature. The ore when introduced into this yields a froth composed of sulphides supported by bubbles of CO_2 . In case the ore is deficient in carbonate, an addition of as much as 3% of calcite or siderite is made. The froth is skimmed with a spoon as soon as it forms. A great deal of mineral is partly lifted, but never reaches the surface. Consequently extractions are low, although the grade of the concentrate obtained is usually very good. For practical purposes, a test of this kind is not of much value, and a better test machine is a small unit shown in Fig. 30. The acid

¹ "Testing Ores for the Flotation Process," *Min. and Sci. Press*, Jan. 1, 1916, the basis of their work being taken from Hoover's original book.

should be allowed to run down through a section of garden hose to within an inch of the surface of the ore. The ore should be kept stirred with a wooden paddle so that the bubbles of CO_2 generated by the action of the acid can lift the sulphides out of the body of the pulp. The froth formed should be skimmed with the paddle as fast as made, then filtered, dried, weighed, and analyzed. Not many ores yield good extractions by this treatment, and the slime gives particularly poor extractions. Fines and Wilfley-table middlings are better adapted, and the presence of siderite in the pulp is desirable, as it reacts slowly with dilute acid. From 1 to 3% sulphuric acid is best in testing, and $\frac{1}{2}$ to $1\frac{1}{2}$ % solutions on the large scale will give about the same results. The temperature of the pulp should be maintained at 70°C . by use of a steam jet. Five to ten pounds of ore per test is necessary. The extractions obtained are always lower than in full-size units. While oil is not necessary in this process, it will greatly assist in the flotation, and the addition of a small amount is often of much assistance in test work.

Tests Using Coal Tar.—An illustration of the method of making flotation tests, as carried out in some cases, is given by William A. Mueller.¹ In giving this description, which was primarily for the purpose of determining the value of coal tar as a flotation agent, Mueller describes the usual small two-compartment machine which was used. One compartment comprised agitation mechanism, which agitated pulp by means of a stirrer. This agitation compartment was connected with the other compartment—the spitzkasten—by a narrow slit near the bottom and a large opening near the middle. This enabled the aerated pulp to flow from the agitation into the next compartment, where no agitation took place. The froth rose to the surface and was skimmed off with a spoon or spatula. The test was not complete until no more mineral-bearing froth was formed. The capacity of the machine was about 2,000 c.c. In all respects this machine was just about the same as the others which have been described heretofore.

For each experiment the time of flotation was noted, this being the period from the beginning of flotation until no more mineral-bearing froth formed in the machine. In all experiments the first product formed was called concentrate, and when this no longer came off, turpentine was added and the next product was

¹ "Use of Coal Tar in Flotation," *Eng. and Min. Journal*, Oct. 9, 1915.

called middling. The system corresponded to that followed when the ore goes to the Mineral Separation type of machine, where the concentrates are formed in the first two compartments, then turpentine is added, and the latter products taken off as middlings and retreated, either in the same machine or in a similar one.

The Elmore Processes.—Other processes of flotation that have not been described here in the experimental way, are the two Elmore systems—the bulk-oil system and the vacuum process. Neither of these processes is used to any great extent in this country at the present time, so that there is no recent addition to the original literature as to their experimental methods. The description given herewith is taken from Hoover's original book. In that description he shows that the old Elmore process may be tried in a glass jar of about 1,000 c.c. capacity. What is known as a Mason fruit jar with a screw top is the best apparatus for this trial. Into the jar 300 c.c. of water is placed, together with 100 grams of sulphide ore, and the whole stirred to make a uniform pulp. Then $\frac{1}{2}$ c.c. of strong sulphuric acid and 200 c.c. of thick oil are added, the oil being a heavy, viscous oil, such as the thick, tarry residue of petroleum, or perhaps a cylinder oil. Screw on the top of the jar, turn it end for end about twenty times, and then set aside to settle for a few moments. The contents of the jar will settle into layers, and at the bottom a layer of sand mixed with some sulphides will be seen. Next above this will be a layer of muddy water, and above that again will be a layer of oil containing most of the sulphides, the amount depending on the success of the maneuver, together with some gangue. The oil layer may be removed by adding more water carefully beneath the oil through a tube, and after the oil and sulphides have been removed, the sulphides can be separated from the oil by heat and gentle agitation, when they will settle to the bottom. This is, in other words, simply reducing the viscosity of the oil so that the sulphides may be dropped through it.

The new vacuum process is more difficult to test in a small way so as to produce satisfactory results. Probably the best method is as follows: Take a large-mouthed 300-c.c. bottle, and fit a stopper with a tube to be attached to a laboratory suction pump. Put into the bottom 30 gr. of sulphide ore, 100 gr. of water, $\frac{1}{5}$ c.c. of Texas or California fuel oil, $\frac{1}{5}$ c.c. of sulphuric

acid. Stir this with a glass rod until the uniform pulp is produced, but do not beat any air into the pulp. Place a stopper in the bottle and attach to the suction pump. Stir the pulp and gently shake the bottle to keep the ore stirred up a little. As the vacuum is produced, the sulphides should begin to rise to the surface and form a layer of froth. While this process has not been applied to any great extent in this country, at least to my knowledge, it has been applied with successful results in some foreign mills. The Dolcoath tin mines of Cornwall, some mines of the Zinc Corporation, and some of the Norwegian iron mines have used the process successfully.

Recording Test Work.—The matter of record for test work has already been referred to, but a rather good form of recording data is given by Whittaker and Belchic.¹ In pointing out the use of some recording form, the authors call attention to the increasing importance of flotation as a method for ore concentration and metal conservation, which is illustrated by the fact that many private corporations, as well as university laboratories, are engaged in investigating the process. Most of these investigators have as their object the defining of the conditions necessary for successful flotation when applied to particular material, while others are engaged in purely theoretical work, looking toward an explanation of the various phenomena upon which flotation is based. Whatever the object sought, the investigation, if prosecuted to any extent, will result in the accumulation of a large amount of minute data, which unless arranged in such a way as to allow of ready comparison and correlation, may soon become a mass of unwieldy material, difficult of clear interpretation.

In order to compile in a condensed arrangement the results obtained in the laboratory and thus allow quick and easy comparison and correlation, the form illustrated here was evolved and printed in quantity. With this arrangement the investigator has before him on one sheet (1) the screen analysis, (2) the assays, (3) the charge used, (4) the derived data, and (5) the plotted efficiency of the frothing agent. The same form may be used also when the essential other flotation factors are being investigated, such as the variations of temperature, duration and speed of agitation, quantity of acid, etc. This condensed scheme

¹"A Form for the Classification of Flotation Data," *Met. and Chem. Eng.*, Jan. 1, 1916.

Flotation Experiments

Number of Sample 6 Date May - 1915
 Assay { Zinc = 5.17 % Number of Experiment 12
Lead = trace

SCREEN ANALYSIS

	SCREEN		WEIGHTS			ASSAYS	
	inch	Opening inches	Sample Weight grams	Per Cent.	Per Cent. Accumulative Weight	% Zinc	
<i>All Pass.</i>	<i>25.</i>	<i>.0145.</i>	—	—	—	—	—
<i>Retard.</i>	<i>25.</i>	<i>.0145.</i>	<i>2.0</i>	<i>0.417</i>	<i>0.417</i>	<i>5.17</i>	—
"	<i>65.</i>	<i>.0025</i>	<i>5.0</i>	<i>0.485</i>	<i>1.092</i>	<i>5.54</i>	—
"	<i>100.</i>	<i>.0018</i>	<i>8.0</i>	<i>1.875</i>	<i>2.967</i>	—	—
"	<i>150.</i>	<i>.0011</i>	<i>34.0</i>	<i>7.08</i>	<i>9.95</i>	<i>7.8</i>	—
"	<i>200.</i>	<i>.0009</i>	<i>85.0</i>	<i>17.70</i>	<i>27.65</i>	<i>4.58</i>	—
<i>Res. fbrs</i>	<i>200.</i>	<i>.0008</i>	<i>341.0</i>	<i>72.30</i>	—	<i>5.56</i>	—
<i>Totals</i>	—	—	<i>480.0</i>	<i>100.0</i>	—	—	—

To show the effect of variation in quantity Pine Oil

CHARGE		Variable grams	DERIVED DATA			
Ore, grams			Froth	Assay of Con.		% Ext.
	<u>400.</u>			<u>53.2</u>	<u>—</u>	<u>74.6</u>
Water, "	<u>1200.</u>	<u>0.5</u>	<u>29.0</u>	<u>53.2</u>	<u>—</u>	<u>74.6</u>
Acid, " <i>Na₂SO₄</i>	<u>2.0</u>	<u>1.0</u>	<u>38.0</u>	<u>43.1</u>	<u>—</u>	<u>79.2</u>
Oil, "	<u>Variable</u>	<u>1.5</u>	<u>37.0</u>	<u>42.4</u>	<u>—</u>	<u>75.8</u>
Temp. in °C	<u>65</u>	<u>2.0</u>	<u>51.0</u>	<u>28.6</u>	<u>—</u>	<u>70.0</u>
Speed, RPM	<u>2000</u>					
Time, minutes	<u>7</u>					

EFFICIENCY CURVE

Efficiency of Pine Oil in terms of frothy assay and extraction.

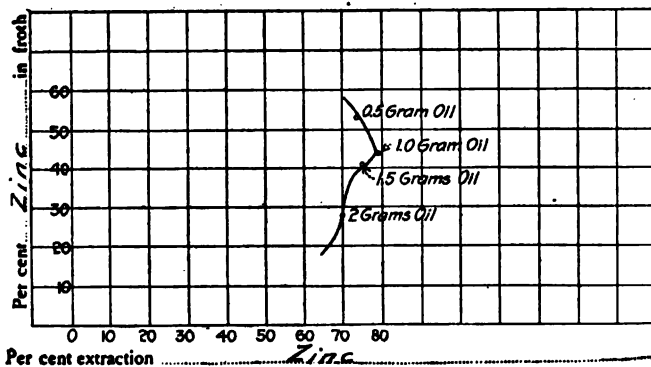


FIG. 31.—A recording form for flotation experiments.

has worked so satisfactorily that it was thought that its publication might be helpful to those who are beginning investigations on flotation and possibly to those already launched upon the subject. The work was done at the laboratory of the University of Kansas, at Lawrence, Kan.

In a review of what has already been brought out in regard to flotation testing, it may be as well to direct further attention to the fact that the results secured in the laboratory cannot be as satisfactory as results secured by plant work on the large scale. The reasons depend principally upon the smallness of the machine, of the very shallow layer of froth that is often formed under such conditions. Laboratory operations, also, seem inevitably to call for greater quantities of oil, acid, and whatever other chemicals are used to secure results. It seems impossible to duplicate the very small quantities, of oil for instance, used in practical work on the plant, in laboratory experiments.

Of the machines that have been described, and the methods accompanying them, there can be no recommendation in general as to which is the best. In this country the oil flotation—with or without acid, but more generally without it—holds the greatest amount of favorable consideration. The surface-film methods such as the Macquisten and the Wood, have been tried and are really being used in practice. They give good results under some circumstances, but in other cases the oil-froth processes seem to be better. It is believed in general that the latter processes are easier to install, apply, and manage, since they seem to require less delicate regulation than the film methods. As to such processes as the Elmore, both old and new systems, the Potter-Delprat, and the others already mentioned, their application is exceedingly limited, at least in the United States, and one may say in North America in general.

Floating Oxidized Ores.—Considerable work on the flotation of oxidized ores has been done by the Salt Lake Station of the Bureau of Mines in coöperation with the metallurgical department of the University of Utah. An article on the subject has been written by O. C. Ralston and Glenn L. Allen, and issued by the Bureau of Mines. Attention is called to the fact that most of the experimental work has been done on oxidized ores of lead, because little success has been had with

zinc ores, and many others are engaged in working on copper ores.

The successful flotation of oxidized mineral depends, in the opinion of the operators, upon its preliminary sulphidizing, in order to convert at least the surface of the mineral particles to sulphide form. This step is then followed by flotation just as is practised on the natural sulphide ores. The sulphidizing methods investigated have been that by the use of hydrogen sulphide on either dry or wet ore; by the use of various sulphide solutions and sulpho compounds of sodium; by the use of solutions of the various sulphides and sulpho compounds of calcium; by the use of sulphur vapor; by the use of a sulphureted oil, and with colloidal sulphur. It was found that some of these methods formed a sulphide film on the surface of the particles of such minerals as lead carbonate or copper carbonate, whereas in other cases the mineral particles are sulphidized all the way through.

The methods have been tried on a large number of lead-carbonate ores. Some contain silver and some lead as the principal metal. Many ores have been successfully concentrated, while others refused to yield to flotation concentration. In general, a high alumina content (acid soluble) seems to prevent sulphidizing and flotation. It was found that the best method of applying hydrogen-sulphide gas to dry powdered ore was in a tumbling barrel with a gas inlet in the end. Sulphidizing in a glass bottle showed that the ore blackened quickly after the application of the gas. On attempting to float the lead sulphide as soon as it was blackened, it was found that a low extraction was obtained and also low-grade concentrates unless the pulp was previously acidified with sulphuric acid. By acidifying the pulp, cleaner concentrates were secured, but extractions of lead still remained low. Only by prolonged treatment with hydrogen sulphide gas could the extraction of lead be raised to commercial grade. With a number of ores 8-hr. treatment gave an extraction of over 80% of the lead.

Hydrogen sulphide was considered because it can be cheaply generated. With iron matte available at \$5 to \$10 per ton and sulphuric acid at about the same price, the cost of the hydrogen sulphide resulting, including all operations, is between \$30 and \$50 per ton. If this gas in combining with the metal in the ore produces only a superficial film of sulphide, it might be

possible to make a ton of gas sulphidize many tons of ore. It was found, however, that the gas attacked the ore with such avidity that by the time the latter was sulphidized sufficiently to permit of good extraction by flotation, it had been sulphidized all through, and practically a chemical equivalent of hydrogen sulphide to the lead in the ore had been absorbed. Application of hydrogen sulphide to the ground ore suspended in water does not seem to be subject to the same difficulty. Filming of the particles seems to take place, and the extractions after a short treatment with the gas are satisfactory.

The best results on lead-carbonate ores were obtained when sodium sulphide was used for the sulphidizing agent. Sodium sulphide has to be introduced in solution and seems to cause true filming. The sulphides of sodium considered commercially applicable are the normal sulphides, Na_2S ; sodium polysulphides, Na_2S_4 , and Na_2S_5 , and the sulphhydrate of sodium, NaSH . The last seems to be quite effective. The normal sulphide is almost as effective, while the polysulphides are the least active. Different ores require from 10 min. to 24 hr. of contact with the sulphide solutions used, depending on the characters of the ore and the substance of the solution. Sodium sulphide varying from 10 to 20 lb. per ton of ore is usually sufficient and should be applied to pulp containing about 1 ton of water per ton of ore, in order that the solution may be as strong as possible during the sulphidizing stage of the process. After a good black color has developed and the color has ceased to increase in blackness, the pulp is diluted with water to 3:1 or 4:1, and floated. The market for sodium sulphide is limited, and it should be obtainable for less than 2c. per pound, making a process economical enough for commercial purposes.

The polysulphide of calcium, obtained by boiling powdered sulphur with slacked lime, is satisfactory for ores that yield easily to sulphidizing, but is sluggish in action as compared with the sulphides of sodium. Sulphidizing with sulphur vapor has been tried with little success, for the reason that it must be applied at a temperature above the boiling point of sulphur in order to prevent condensation. This means that a temperature of over 445°C . must be reached. The use of a sulphureted flotation oil, in which loosely combined sulphur is available for combination with carbonates of lead or other metals, and the rest of the oil is then available for oiling the artificial sulphide,

has given but little encouragement. Colloidal sulphur, mentioned as a possible method of sulphidizing, does not seem to combine with lead carbonate at all. It floats as a white lining of air bubbles in the flotation machine and brings up very little lead with it.

The Course of the Precious Metals.—Usually the precious metals contained in a lead-carbonate ore accompany the lead. It has been noticed that the silver extraction will lag behind the lead extraction when the ore is sulphidized with sodium sulphide and that the reverse has usually been true when hydrogen sulphide was used. The importance of sulphidizing flotation is due to the fact that there are many deposits of lead-carbonate ore in the Western States, and many of these ores have been milled with varying success. Frequently lead carbonates can be satisfactorily concentrated by gravity methods, but often it is found that particles of lead carbonates go into the slime and are lost. Tailings containing 5 to 10% of lead are common, and the object of the investigation was to determine whether sulphidizing flotation could be applied to the treatment of these deposits, and also to prevent the waste that now takes place when they are treated by gravity concentration. It would also, if successful, render amenable to treatment carbonate ores that are too low in grade to be handled by present methods. A flotation plant to apply sulphidizing flotation to an ore containing lead, silver, and gold was constructed by the Prince Consolidated Mining Co., at Pioche, Nev., for the treatment of two tailings dumps from former pan amalgamation and cyanide operations in that vicinity.

Effect of Pulp Dilution.—The effect of the dilution of a pulp that is subjected to flotation is a very important factor in practice, and therefore should be considered in detail in any complete system of testing. In this connection, Oliver C. Ralston, in Technical Paper No. 149 of the U. S. Bureau of Mines, June, 1917, says:

If a given ore be crushed to a definite size and treated by flotation, the more dilute the pulp, the cleaner is the concentrate.

This is because less gangue is entrained in the froth made from a more dilute pulp. It is also noted that the extraction also falls off, mainly because the coarser particles are not floated. It would, seem that in the more concentrated pulp the smaller

particles of mineral assist the floating of the larger particles. When flotation is attempted in dilute pulps, these larger particles are not as close to their smaller neighbors and are not carried into the froth with them. If the ore is already finely ground, the thicker pulp often gives such a poor concentrate that dilution greatly increases the grade of the concentrate. In fact, some slimes cannot be treated in any other than a dilute pulp—5 or more tons of water to 1 ton of ore. These slimes are usually from ores with a gangue containing much colloidal matter.

Another effect noticed on treating pulps of different dilutions, is that dilute pulps require more oil for a good froth than do the thicker pulps. Of course, there is a greater weight of water to the oil, and likewise, there is more water to be agitated or otherwise treated in order to form a froth. This means a greater expenditure of power per ton of ore treated.

Sand and Slime.—From what has already been said it is evident that granular matter can be treated in a very thick pulp, whereas the successful treatment of slimes usually requires more dilute pulp. The surface presented by a ton of slime is immensely larger than the surface of the particles in a ton of sand, so that more oil is needed in treating slime. Furthermore, when the material is very finely divided, it usually tends to gather together in flocks, in some of which a number of gangue particles may surround each sulphide particle. Such flocules must be broken up in one way or another in order that the sulphides can be oiled and floated. One way of accomplishing this is by extreme agitation, and another is to deflocculate the gangue slime by the addition of a deflocculant, such as caustic soda, sodium silicate or a similar agent. Because of the necessity of floating the slimes in a more dilute pulp, the greater amount of oil, power, water, etc., involved, makes the treatment more expensive. The use of a dilute pulp cuts down the capacity of a flotation machine as far as the tonnage of ore is concerned, and sometimes it even cuts down the total volume of water and ore together put through the machine. Thus one can say in general that a given machine will have a greater capacity for granular matter than for slime.

A true colloidal slime resulting from pulverization of the clay constituents of the ore or gangue, often proves difficult to treat by flotation. In this connection, the work of the metallurgists at the mill of the Inspiration Consolidated Copper Co. has been

most interesting. The primary slime produced at that mill could not be treated successfully by itself. Addition of iron filings made possible the formation of a good mineral froth, and later it was found that a certain portion of granular matter in the slime gave the same effect. In other words, it is not best to remove granular matter completely from slimes not usually amenable to flotation, as the mere presence of the proper percentage of granular material will permit successful flotation of the total pulp.

Mr. Ralston, in his paper, points out that no such satisfactory explanation for this result is known. While it is true that none has been universally agreed upon, it may be pointed out that the process tending to flocculation that has been already mentioned in the chapter on the theory of flotation, probably has a good deal to do with the matter. It has been suggested that electrostatic charges upon particles in solution or water may deter or expedite flocculation. I believe it has been definitely shown that conditions tending toward a high degree of dispersion are obstructive to good flotation results, and that, on the other hand, conditions that tend to break down emulsions are contributory to satisfactory results. It has also been demonstrated, I believe, that in cases where flotation was not obtainable on finely ground material dispersed in water, that the introduction of some means to discharge the particles, permitted flocculation and consequent flotation.

Handling Coarse Material.—In speaking of the matter of handling coarse material, Mr. Ralston, in the paper already referred to, says that if the coarse material is contained in a pulp formed by simple grinding of the ore, without classification or removal of the slime, it is easier to float some of the coarser material than when the product has been de-slimed. The coarse is entrained in flocules of smaller material. It can also be seen when a dilute pulp should not be used. The pulp should be as thick as possible, so that the coarse particles of ore cannot drop through the gangue particles and the water too rapidly. The frothing agents that give a tough rather than a brittle froth will also be of advantage, and the flotation machine used should be capable of making tough froth, that is, a mechanical broken frothing machine. These, in the main, constitute the conditions for successful flotation of coarsely ground ore.

As to the desirability of floating such material, the gangue

minerals and the sulphides may be of so nearly the same specific gravity that gravity concentration would be difficult. Middling formed in a gravity concentration mill treating such ore, would not be amenable to froth flotation on account of the lack of slime. However, mixing such middling with the slime that needs flotation, it could yield a mixture adapted to froth flotation, unless the middling were very poor. Even then it might require a small amount of grinding, but not as much as would be needed if froth flotation of the middling alone were to be attempted. Where the gangue minerals are not heavy, the coarse ore in a pulp can be recovered on tables, and the necessity for its flotation need not be great. It is evident that flotation of all the coarse ore would make unnecessary its recovery by concentrating tables, but up to date the possibility of doing this has not been demonstrated. The necessity of floating coarse mineral comes usually at times when the ordinary grinding capacity of a mill has been less, through a breakdown or other causes. Under these conditions, if pulp containing some relatively coarse material must be sent to the flotation machines, the mill man will find it necessary to alter some of the factors in order to adapt the flotation conditions to the pulp at hand. In case it is wished to crowd production at such a mill, the grinding machinery probably would be adjusted to send coarser material to the flotation machines. As flotation machines have a much greater capacity for sands than for sands and slimes, and as sand assists the flotation of the slimes, it will be found that such crowding of a mill will not cause very serious losses of valuable minerals in the tailings. The success of the Inspiration Consolidated Copper Co. in obtaining high capacity from its machines, has fairly well demonstrated the truth of this reasoning. From 2 to 6% of the flotation-machine feed at the Inspiration mill will not pass a 48-mesh screen, and although the sand tailing from the machines has to be concentrated on tables, the total recovery of valuable minerals by these tables is small, being only 10% of the total mill concentrates. As gravity concentration can make fair recoveries from 200-mesh sand by gravity concentration, it can be seen that in this case, flotation is replacing gravity treatment of sand to some extent. At Inspiration the main reason for the procedure mentioned was to provide enough granular matter to assist the flotation of the refractory primary slimes.

Tests on Copper and Zinc Ores.—A few tests were made on oxidized copper ores, and hydrogen sulphide seems to be the best medium for sulphidizing these particles. When applied to the dry ore, the sulphur permeates the particles as it did in the lead ores. Applied to the wet pulp, the hydrogen sulphide seems to cause true filming. Usually black concentrates are yielded, but J. M. Callow, of the General Engineering Co., announced that that company has been able to reduce the amount of sulphur used to a point where the froth is green with slightly coated malachite. He has used as little as $\frac{1}{2}$ lb. of sulphur per ton of ore, and that is giving good extraction in the plant of the Magma Copper Co., at Magma, Ariz., where his company has put in a successful installation. Sodium sulphide has been tested by a number of large companies where some little oxidized copper occurs in the sulphide ores. In this case only 2 or 3 lb. of sodium sulphide per ton of ore is used. This is easily added to the machines during flotation or to the mixing tanks before flotation. The experience is that if some little time of preliminary contact is allowed before flotation is attempted, better sulphidizing of the material will result. Calcium polysulphide has been used for some time in a number of large copper mills with indifferent success and seems to be detrimental in some instances. On some of the ores tested fair results were obtained if the calcium polysulphide was allowed to act until the ore had become well blackened. Sulphur vapor is also said to give good results in some cases. Sulphureted oils are being used also, but considerable secrecy is observed as to technical details of the work. Colloidal sulphur does not assist, so far as is known, in the flotation of oxidized copper. Attempts to float the natural sulphides along with sulphidized minerals have usually failed, since the sulphidizing agents cause trouble with the flotation of natural sulphides. At one plant this difficulty has been solved, although the details are not available.

Attempts to float oxidized particles of zinc from ores, both before and after sulphidizing by most of the methods that have been mentioned, have met with no success whatever in the laboratory experiments carried out. Possibly the fact that most of the carbonate ores of zinc contain important amounts of the silicate may be one of the reasons for the lack of success with this work.

In testing according to the directions and indications that have been given here, almost exclusive attention has been given to the

base metal ores, and little has been said about gold and silver. As a matter of fact, there has been little public admission that the process has been applied to any such ores, but cases are privately known where free gold has been successfully recovered by flotation, and there are also cases that definitely show that silver sulphides, such as occur largely in Mexico, may easily be recovered by flotation. The indications are strongly in favor, in the latter case, of separating the ore into sands and slimes, concentrating the former by gravity methods, and the latter by flotation methods. When making flotation experiments upon ore of this kind, this fact should be borne in mind.

CHAPTER VII

TESTING AT THE ANACONDA MILL

As a practical example of test work carefully carried out, nothing better could be cited than that performed at the Washoe plant of the Anaconda Copper Mining Co. The procedure has been described by Frederick Laist and Albert E. Wiggin in the Bulletin of the American Institute of Mining Engineers, March, 1916. The value of this work is so great that I have considered it well worth a special chapter in this book, so that a review of the work done will be given here.

Early in 1914 it was decided to test, on a large scale, the treatment of flotation of Anaconda slime and mill tailing. For this purpose a standard type Minerals Separation machine was installed at the Washoe Reduction Works during May and June, 1914. This was followed by the installation of a full-size Callow pneumatic plant. In addition to the tests made in the standard type Minerals Separation machine some tests were made using a Minerals Separation machine of the sub-aeration type.

During the series of experiments a large variety of oils was tested. Experiments were also conducted using both round-table feed and tailing to determine whether it would be better to displace the round tables by flotation for the treatment of the slime, or to supplement the round tables by flotation of the round-table tailing. A series of tests was also made on the treatment of the mill tailing by grinding followed by flotation to determine the relative merits of flotation and leaching for the treatment of this product. In addition, flotation tests were made on mixtures of mill tailing and slime.

The round-table feed referred to above is the total slime from the mill. It contains about 35% colloidal solids and approximately 90 to 95% of the total solids will pass through 200 mesh (0.067 mm.). It assays from 2.3 to 2.6% Cu. The mill tailing referred to above is the total discard from the mill, exclusive of the slime. It is all finer than 2 mm. and about 90 to 95% will remain on 0.25 mm. It assays about 0.06% Cu.

Preliminary Tests.—A series of tests was first carried out to determine roughly the best conditions for flotation, using the standard Minerals Separation machine and treating round-table feed. The following reagents were tested either alone, or in combinations: Turpentine, crude oil, cresylic acid, stove oil, tar oil, Carolina oil of tar, argols, sludge acid, fuel oil, wood creosote, and sulphuric acid. In some of these tests sulphuric acid was used and in others it was omitted. The effect of the temperature of the pulp upon the flotation results was tested by heating to various temperatures. As these tests were merely preliminary no record was kept of the amount of reagents used. It was conclusively proved, however, that the best combination of reagents was sludge acid, wood creosote, stove oil, and sulphuric acid. Of all the reagents tested, these happened to be the cheapest. It was also proved that the addition of sulphuric acid to the pulp was of decided advantage in the treatment of slime. In two successive tests in which sludge acid, wood creosote, and stove oil were used, the tailing assayed 1.25% Cu when no acid was used and 0.3% Cu when acid was used. Since these tests were made we have omitted the use of stove oil.

This machine had 16 agitator compartments, each 2 ft. square, and 14 spitzkastens, and was of the standard Minerals Separation design. This machine is known as M. S. Machine No. 1. The agitators were of the standard Minerals Separation type, the impellers being 18 in. in diameter and the agitators making 265 r.p.m. This gave the impellers a peripheral speed of 1,245 ft. per minute. The machine required 45 to 55 hp., including motor and belt transmission loss, when operating under a full load of slime pulp. The first products to be tested were the round-table feed and tailing.

(a) *Round-Table Feed.* Normal Tonnage (60 Tons). Pulp Heated to 70° to 90°F.—The reagents used in Periods 2 and 11 were kerosene sludge acid (M. S. 37), crude wood creosote (M. S. 33), stove oil (M. S. 8), and sulphuric acid.

Period 2, in which the pulp was heated to an average temperature of 89° F., gave a tailing running 0.23% Cu, with a fair grade of concentrate. Period 11, in which more creosote was used, all other conditions remaining as in Period 2, gave a tailing assaying 0.27% Cu, with a slightly better grade of concentrate. It should be noted, however, that it is not characteristic of creosote to give a clean concentrate, it being rather a good

"tailing" oil and the slight increase in the grade of concentrate should not be attributed to the increase in the amount of creosote used, but rather to a difference in operation of the machine. Period 20, in which Pensacola creosote was used, shows an average tailing of 0.33% Cu, and a very clean concentrate. It should be noted that considerably less creosote was used than in Period 11.

(b) *Round-Table Feed. Pulp Not Heated.*—During Periods 3 and 10, the pulp was not heated, but treated at a temperature of 50°F. The tests were made during the summer months; it should be noted that during the winter months the temperature of the pulp would be about 40°F., with a minimum of about 35°F. During Period 3, the tailing assayed 0.32% Cu, with a low-grade concentrate, and during Period 10 the tailing averaged 0.42% Cu, with a higher concentrate.

(c) *Round-Table Feed. Pulp Heated to 125°F.*—Heating the pulp to a temperature of 125°F. (Period 4) gave a tailing of 0.21% Cu with a fair grade of concentrate. As Period 2, in which the pulp was heated to 90°F., gave a 0.23% Cu tailing it is evident that heating beyond 90°F. does not pay. In fact, a careful study of the tabulated results would indicate that it does not pay to heat above about 75°F. The cost of heating the slime pulp for flotation is about 1c. per ton of dry slime for every 2½°F.

(d) *Round-Table Tailing. Pulp Not Heated.*—The treatment of the round-table tailing, after dewatering to a density of 11.5% solids (Period 5), gave a tailing of 0.25% Cu with a low-grade concentrate. It was thought that possibly the net recovery would be sufficiently more to justify the retention of the round tables if the tailing from the round table instead of the feed to the round table were treated by flotation.

(e) *Round-Table Tailing. Pulp Heated to 106°F.*—It is evident from the result of this test (Period 6) that the heating in the case of the round-table tailing was of no benefit.

(f) *Round-Table Feed. High Density of Pulp. Pulp Heated to 96°F.*—The density of the round-table feed was increased to 22.5% solids by a second dewatering (Period 7). As had been expected, the increase in density considerably decreased the oil consumption without decreasing the recovery of copper, and of course would also decrease the expense of heating the pulp.

(g) *Round-Table Feed. High Density of Pulp. High Tonnage (105 Tons). Pulp Heated to 96°F.*—It was thought that by increasing the density of the pulp it might be possible to treat the same volume of pulp per machine unit as with the lower density, and in that way increase the tonnage of solids treated. The tests under Period 8, however, show that any considerable increase in the amount of feed to the Minerals Separation No. 1 machine over 60 tons results in a decided increase in the tailing loss.

(h) *Round-Table Feed. High Density of Pulp. High Tonnage. Pulp Not Heated.*—The value of heating the pulp in the case of high density feed pulp is brought out by this test (Period 9), where the tailing jumped to 1.28% Cu with the absence of heat.

(i) *Round-Table Tailing. High Pulp Density. Pulp Heated to 70°F.*—This test (Period 12) gave a tailing of 0.45% Cu, and a low-grade concentrate.

(j) *Round-Table Tailing. Medium Pulp Density. Pulp Heated to 80°F.*—This test (Period 13A) gave a good tailing assaying but 0.20% Cu. This would seem to indicate that too high a pulp density in the treatment of the round-table tailing is a decided disadvantage.

(k) *Round-Table Tailing. Medium Pulp Density. Pulp Heated to 80°F.*—In this test (Period 13B) no acid was used, and as a result the tailing went up to 0.96% Cu. Note also the low silver content of the concentrate.

(l) *Round-Table Tailing. Low Pulp Density. Pulp Heated to 70° to 80°F.*—In (Periods 14A, B, C, and D) the round-table tailing pulp was not dewatered at all, but treated directly at a density of about 6% solids. The tailing was good, ranging from 0.16% Cu to 0.22% Cu, for the four tests. During Period 4B, crude turpentine was used in place of creosote; during Period 14C, crude wood creosote was used, and during Period 14D, crude wood creosote was used. All of these oils gave satisfactory results. The increased amounts of these oils used during these tests were undoubtedly due to the low density of the pulp, and to the low tonnage treated.

(m) *Round-Table Feed. Low Pulp Density. Low Tonnage (40 to 50 Tons). Pulp Heated to from 70° to 80°F.*—It was thought that the low tailing produced during Periods 14A, B, C, and D was due to the low tonnage treated, and possibly to the

low pulp density, as well as to the fact that round-table tailing was being treated. Accordingly, the tests under Periods 15A, B, C, D, and E were made and in these a low tonnage of round-table feed was treated with a low pulp density. These tests gave fully as good results from the recovery standpoint in a considerably better grade of concentrate as those made on the round-table tailing under similar conditions.

During Period 15A, wood creosote was used. During Period 15D, no creosote was used and the tailing assay increased from about 0.20% Cu to 0.32% Cu. During Period 15E, some other creosote was used, but gave a high tailing—0.42% Cu. During Period 15B, tar creosote from a gas works was used with apparently good results, although the test was too short to be taken as conclusive proof of the value of this creosote as a flotation oil.

(n) *Round-Table Feed. Low Tonnage (59 Tons). Pulp Heated to 70°F.*—The tests (Period 16A) gave a tailing running 0.21% Cu, and a fair grade of concentrate.

(o) *Round-Table Feed. Pulp Heated to 70°F. High Speed of Agitators.*—It was thought that by increasing the speed of the agitators the amount of feed treated might be increased without decreasing the recovery. The first nine agitators were used and their speed was increased from 265 r.p.m. to 363 r.p.m. The test (Period 16B) was of short duration, as the machine would have racked itself to pieces if operated very long at this high speed. While the test is too short to be conclusive, the indications are that the increased speed was a detriment rather than an advantage. It is thought that there is a critical speed at which the agitators should be run for the most efficient operation. If we go above this speed the power increases much faster than the capacity, and if we go below it we do not get a proper mixing of oil and acid with the pulp. With the agitators running at this speed, the nine cells required 56 hp. including motor and belt transmission loss.

(p) *Round-Table Tailing. Low Tonnage (40 Tons). Pulp Heated to 70°F. Low Pulp Density.*—These tests (Periods 17 and 18) show average tailings of 0.22% Cu and 0.26% Cu respectively. The concentrate is rather low grade.

(q) *Round-Table Feed. High Tonnage (90 Tons). Pulp Heated to 70°F.*—The test under Period 19 gave a high tailing, 0.43% Cu, and a good grade of concentrate. The high tailing is

due entirely to overloading the machine. Under Period 23 the tonnage treated was just as high, the grade of concentrate only a little lower, and yet the tailing is good, 0.25% Cu. This apparent discrepancy between these two tests is explained, it is thought, by the fact that the slime produced in the mill was considerably more granular during the tests under Period 23 than during those under Period 19.

(*r*) *Round-Table Feed. Low Tonnage (40 Tons). Pulp Heated to 70°F.*—In this test (Period 21), the tailing averaged 0.25% Cu and the concentrate was medium grade. The tailing is practically no better with this low tonnage (40 tons) than under Period 23 with the high tonnage (90 tons).

(*s*) *Round-Table Feed. High Tonnage (90 Tons). Air Used in Last Spitzkasten.*—It was thought that the introduction of air in the bottom of the last spitzkasten might increase the capacity of the machine and at the same time maintain the same mineral recovery. At first it seemed as though the introduction of the air (Period 22) had resulted in a considerable increase in the capacity of the machine, but when the tests under Period 23 were made, in which the air was omitted, it was found that it was not the introduction of the air which had increased the tonnage.

(*t*) *Conclusions.*—1. The economic capacity of the Minerals Separation No.1 machine when treating slime as produced from the mill, May, 1915, seems to be from 80 to 90 tons.

2. The best combination of reagents for the treatment of slime seems to be sulphuric acid, kerosene sludge acid, wood creosote, and stove oil. There is some question as to the real value of the stove oil. Its principal function seems to be to make a more compact froth.

3. It would not be economical to retain the round tables, as the recovery by treating the slime directly by flotation is just as high as by retaining the round tables and treating the round-table tailing by flotation. The grade of concentrate would probably be the same in either case, but any difference would be in favor of treating the round-table feed directly by flotation. The heating of the round-table tailing pulp, on account of its low density, would increase the cost of the flotation.

4. In treating the round-table feed directly by flotation, the resulting tailing should assay 0.30% Cu, or less, with a concen-

trate carrying not over 40% insoluble. Possibly the concentrate can be made much cleaner with no sacrifice in the recovery.

5. It is thought that the best circuit density for the slime pulp for flotation treatment is about 12% solids.

6. It is thought that about 70°F. will be found to be the most economical temperature at which to keep the pulp.

7. Acid seems to be absolutely essential to the successful treatment by flotation of the slime.

8. The addition of air in the last spitzkasten is of no advantage.

9. Any considerable increase in speed of the agitators above a peripheral speed of about 1,300 ft. per minute seems to be disadvantageous.

Treatment of Mill Tailing after Grinding through 60 Mesh.—

These tests were made in the Minerals Separation No. 1 machine. Mill tailing from Sections 7 and 8 of the concentrator were elevated and then dewatered. The dewatered tailing was then crushed through 60 mesh (0.25 mm.), either in a Hardinge mill, 10 by 4 ft., or a tube mill, 8 by 12 ft. The grinding mills were operated in closed circuit with a Dorr classifier, the overflow of the classifier being the final product of the system and going to the flotation plant for treatment.

Accompanying is a screen sizing test of the average final product produced during the test on the Hardinge mill, and is typical of the flotation feed:

(a) *Preliminary Tests.*—These tests (Period 24) were started immediately after putting the Hardinge mill in operation. At first no sulphuric acid was added, and the pulp was not heated. Kerosene sludge acid contains from 50 to 60% sulphuric acid, so that when this oil is used we could not have a non-acid pulp. We found, however, that the use of acid in addition to that contained in the sludge was of advantage. Some very low tailings were produced during these preliminary tests, but the concentrate was very low in grade. It seemed to be of decided advantage to add the oil ahead of the grinding mill, the latter making an ideal agitator. However, as soon as we began to heat the pulp ahead of the mill, we found that the acid in the oil and the small amount of copper sulphate formed began to corrode the iron work of the mill. This would not be a serious matter if we used a pebble, or siliceous block lining, with pebbles for grinding, but we early abandoned the idea of using siliceous, or pebble linings, and there was a good chance that as a grinding medium we would

find iron balls superior to pebbles. This, of course, precluded the use of any acid ahead of the grinding mill, especially if the pulp was heated.

SCREEN-SIZING TEST ON DORR CLASSIFIER OVERFLOW, OR FEED TO FLOTATION MACHINE

Screen size		Cumulative per cent., solids
Square mesh	Aperture, mm. square	
+16	1.180	0.3
+24	0.730	1.3
+40	0.430	3.0
+60	0.260	5.8
+80	0.210	12.8
+110	0.130	38.5
+130	0.110	42.3
+160	0.085	54.8
+200	0.076	59.3
+240	0.063	62.8
-240	0.063	37.2

It is barely possible that we might find the action of the kerosene sludge acid in a cold circuit so slight that it could be neglected, even though we used steel lining and steel balls. It would be a decided advantage to add the oil ahead of the grinder.

(b) *Low Tonnage (125 Tons). Pulp Heated to 70°F. Oil and Acid Added in Minerals Separation Machine. Acid Circuit.*—The tailing produced during this period (No. 25) averaged 0.07% Cu. The concentrate was too siliceous, averaging 40.1% insoluble. The only reagents used were sulphuric acid and kerosene sludge acid. It should be noted that the product treated during this period contained only 3.1% on 80 mesh (0.20 mm.).

(c) *High Tonnage (187 Tons). Same Conditions as under (b),* except that Flotation Machine was Divided into Two Parts, Making a Primary and a Secondary Machine, the Secondary Machine being used to Clean the Concentrate from the Primary Machine.—The flow sheet was as follows: The feed pulp was put into the sixth agitator cell, thus making the last 11 cells act as a primary machine. The concentrate from the first six boxes of the primary machine was returned to the first three boxes of the original machine through the two pre-agitators. The overflow from the

remaining five boxes of the primary machine was returned to the original feed. The last cell made the usual tailing. The secondary machine made a final concentrate and a middling which was returned to the circuit. The chief reagents used were kerosene sludge acid and sulphuric acid. Small amounts of creosote were used during part of the test, but it seemed to be of no particular advantage.

The tailing during this period (No. 26) averaged 0.11% Cu, and the concentrate averaged 26.9% insoluble. It is thought that this is about typical of the results we may expect from a commercial installation. It should be noted that 14.4% of the flotation feed remained on an 80-mesh screen (0.2 mm.). Finer grinding would undoubtedly result in a lower tailing assay, but we must, of course, set against the increased copper recovery the cost of the finer grinding.

It would be well to call attention to the fact that all averages shown in these flotation tabulations are arithmetical and not geometrical. It was found that the arithmetical average practically checked the geometrical in the test cases, and as the geometrical averages would have involved a great deal of extra work it was considered not worth while to average the results geometrically. For example, Period 26 was averaged geometrically and gave a tailing assay of 0.109% Cu against the arithmetical average of 0.11% Cu. The concentrate averaged 7.777% Cu geometrically, and 7.82% Cu arithmetically. As the daily variations in tonnage and assays are considerable during this period, we would expect to find the arithmetical and geometrical averages diverging more than during the other periods in which the variations are not so great.

(d) *Low Tonnage* (124 Tons). Same Conditions as under "c," except Turpentine (M. S. 14) added ahead of Grinding Mill, and no Sludge Acid Kerosene used. Pulp Heated to about 70°F.—In this test (Period 27) it was endeavored to take advantage of the grinding mill as an agitator and for this reason a neutral reagent, turpentine, was used. The tailing is high, 0.12% Cu, considering the comparatively low tonnage of feed and the fact that only 5.2% of the feed was coarser than 80 mesh. The same flow sheet was used as during Period 26.

(e) *Low Tonnage* (106 Tons). Oil added ahead of Grinding Mill. Non-Acid Circuit. Pulp Heated to about 75°F.—Various mixtures of crude turpentine, creosote, and pine oil used during

these tests. The mixtures contained from 60 to 70% crude turpentine; 21 to 32.5% creosote; 2.5 to 15% pine oil. The tailing produced during this period was fairly good, averaging 0.09% Cu, but the grade of concentrate was not good, averaging 41.5% insoluble.

(f) *Conclusions.*—1. Although not definitely demonstrated, it is thought that the economical capacity of the Minerals Separation No. 1 machine when treating sand tailing crushed through 60 mesh is about 175 to 200 tons per 24 hours.

2. The best combination of reagents seems to be kerosene sludge acid and sulphuric acid. However, a mixture of creosote, turpentine, and pine oil in a non-acid circuit gave good results also. The non-acid circuit, however, seems to require more delicate adjustment and more careful attendance than the acid circuit.

3. The grinding mill makes an ideal agitator, and it is of decided advantage to add the oil ahead of the grinders.

4. The treatment of the mill sand tailing ground through 60 mesh should result in a tailing assaying not over 0.10% Cu and a concentrate carrying not over 30% insoluble.

5. It is thought that the best density of pulp is from 25 to 30% solids.

6. Heating of the pulp to about 70°F. seems to be of advantage, although there is a possibility that this heating may be dispensed with during the summer months without any injurious results.

7. Acid seems to be beneficial but it is not of as much importance as in the treatment of the slime.

Treatment of Mixture of Round-Table Feed and Mill Tailing after Grinding through 60 Mesh.—These tests were made in the Minerals Separation No. 1 machine. It was thought that it might be of advantage to mix the slime and the reground mill tailing for flotation treatment. The kerosene sludge acid, turpentine, and the sulphuric acid used were added in the flotation machine. In some instances, various mixtures of coal tar (70 to 80%), creosote (17.5 to 22.5%), and pine oil (2.5 to 7.5%) were used with the sludge acid. These were added ahead of the grinding mill.

The average proportion of sand tailing to slime in the mixture treated was 75.7 to 20.1, or 3.8 to 1. In practice the proportion of production of tailing to slime is about 3 to 1; thus the mix-

ture was somewhat deficient in slime. The concentrate produced, 34.1% insoluble, is of a good grade, but the tailing is high, 0.20% Cu. Theoretically, the tailing should have assayed about 0.15% Cu, assuming a 0.10% Cu tailing from the sand tailing and 0.30% Cu tailing from the slime.

Although this test was not conclusive, it was decided, from observation, that it is better to treat the slime and the sand tailing separately. Of course, the slime made in the grinding of the sand tailing is included in the sand tailing for treatment. This slime produced in grinding the tailing is much lower grade and more siliceous than the original mill slime.

Tests with Callow Pneumatic Machine.—Tests made by Mr. Callow at his laboratory in Salt Lake on samples of our mill tailing ground through 40, 60, and 80 mesh, and of our slime, had given such promising results that it was decided to try out the Callow machine on a commercial scale. Accordingly, there was shipped here during September, 1914, five standard Callow cells, 2 by 8 ft., a Pachuca agitating tank and accessory apparatus, consisting of blower and sand pumps. This equipment was installed in the old 80-ton experimetal leaching plant and was ready for operation the latter part of October.

In addition to the Pachuca agitator recommended by Mr. Callow, there was built a set of two mechanical agitators. These agitators consisted of a tank about 10 ft. long by $2\frac{1}{2}$ ft. wide and $2\frac{1}{2}$ ft. deep, in which revolved a horizontal shaft carrying a set of paddles. These agitators were belt driven from one motor and required a total of 25 to 30 hp., including motor, belt, and countershaft power loss. The agitators seemed to work well and had a combined capacity of about 60 tons of slime per 24 hours.

Power readings made on the blower which furnished the air for the five Callow cells gave the following results:

Air pressure, pounds	Input to motor, hp.	Shafting, hp.	Net to blower, ¹ hp.
4	25.8	8.8	17.0
5	35.4	8.8	26.6
6	47.4	8.8	38.6

¹ Includes motor loss.

Treatment of Round-Table Feed and Tailing.—(a) *Round-Table Feed. No Acid.*—During the first four days air agitation was employed, using the Pachuca tank. With a feed of 60 to 80 tons, the results were very poor. The air agitator did not have sufficient capacity. During the last five days of this period (No. 32) mechanical agitation was used and gave much better results.

(b) *Round-Table Feed. Air Agitation.*—This test (Period 33) which was of 16 days' duration gave an average tailing of 0.35% Cu, and a concentrate carrying 31.6% insoluble. The capacity per cell seems to be about 15 to 20 tons of slime per day. Sludge acid and sulphuric acid, with no creosote or stove oil, seemed to give as good or better results as when creosote was used.

(c) *Round-Table Tailing. Mechanical Agitation.*—This test (Period 34) of 27 days' duration, gave an average tailing of 0.30% Cu, and a concentrate assaying 34.2% insoluble. The average tonnage per cell was 20 tons per day. The principal oil used was kerosene sludge acid, together with sulphuric acid.

(d) *Round-Table Tailing. Mechanical Agitation.*—This test (Period 35), of four days' duration, gave an average tailing of 0.32% Cu, and a final concentrate of about 60% insoluble. (The rougher concentrate for the first two days should be included in the average final concentrate.)

(e) *Conclusions.*—1. On our slime, air agitation is not as satisfactory as mechanical.

2. The capacity of one standard Callow cell is about 15 to 20 tons of slime per day.

3. The Callow machine produces a clean concentrate but does not give as clean a tailing as the Minerals Separation machine.

4. The Callow machine is more sensitive and requires closer attention than the Minerals Separation machine.

5. The cost of repairs would probably be less on the Callow machine than on the Minerals Separation machine. This cost, however, is comparatively small for either machine.

6. The power required per ton treated in the Callow system is just about the same as that required in the Minerals Separation machine.

In all of these tests the original feed was divided among the Callow rougher cells, operating in parallel. As a rule, there was one cleaner cell operating also. When this was operating the

concentrate from the rougher cell went to it, the cleaner making a final concentrate and a middling which was returned to the system. The rougher cells made the final tailing.

Treatment of Mill Tailing after Grinding through 60 Mesh.—

(a) *Preliminary Test.*—During the first few shifts the mechanical agitators at the Callow plant were used, but it was soon found that they were not required—that the grinding mill gave sufficient and thorough agitation.

Kerosene sludge acid was the only oil used during this period, and was added ahead of the grinding mill. Sulphuric acid was added ahead of the flotation cells. The tailing for this period averaged 0.10% Cu, and the concentrate carried an average of 42.2% insoluble. The pulp was heated just ahead of the flotation cells (Period 36).

(b) *Kerosene Sludge Acid Added Ahead of Grinding Mill.*—The flow sheet was modified slightly during Period 27. Instead of sending the entire rough concentrate from the rougher cells to the cleaner cells, only the first half was sent to the cleaner, that coming off the half nearer the tailing end of the machine being returned as middling to the original feed. This resulted in giving the cleaner cell a richer feed and, in turn, enabled the cleaner to make a higher grade concentrate. This period shows some very good results, a tailing assaying 0.11% Cu, and a concentrate carrying 27.9% insoluble. Here, again, is brought out the splendid agitation and mixing obtained in the grinding mill.

(c) *Neutral Circuit. No Acid. Turpentine Added Ahead of Grinding Mill.*—In this test (Period 38) no acid was used. Turpentine was added ahead of the grinder. The tailing averaged 0.16% Cu, and the concentrate carried 35.1% insoluble. It should be noted that in this test tonnage treated per cell was only 58.2 tons as against 75.6 tons in the preceding test.

(d) *Neutral Circuit. Same Conditions as under "c" except Lower Tonnage per Rougher Cell.*—During this period (No. 39) the average tonnage treated per rougher cell was only 39.9 tons. The tailing was high, 0.18% Cu, and the concentrate low grade, 47.5% insoluble.

It is evident from the work in this and the preceding period that turpentine will not give us good results in a neutral circuit.

(e) *Mixtures of Coal Tar, Creosote and Pine Oil Added Ahead of Grinder. Neutral Circuit.*—The tests during Periods 36 and 37 show a tailing of 0.13% Cu, and a low-grade concentrate—

45.3% insoluble. The results are not nearly so good as those obtained in the acid circuit.

(f) *Same Conditions as under Period 40, except the Rougher Cells Operated in Series instead of in Parallel.*—During this test the original feed went to No. 2 rougher and the tailing from No. 2 rougher was fed to No. 1 rougher, the latter making a final tailing. The concentrate from the No. 2 rougher went to the cleaner and that from No. 1 was returned to the feed as middling. The final tailing averaged 0.11% Cu, but the concentrate was rather low grade, running 38.7% insoluble.

(g) *Conclusions.*—1. The capacity of the standard Callow cell when treating ground mill tailing is about 75 tons per day.

2. No other agitation is required if the reagents can be added ahead of the grinding mill.

3. The use of acid seems to be of considerable advantage.

4. On account of utilizing the grinding mill as an agitator the Callow machine requires less power than the Minerals Separation machine.

5. The Callow machine is more sensitive and requires more attention than the Minerals Separation machine.

Soluble Copper in Flotation Tailing.—Samples of tailing pulp were sent to the laboratory and the soluble copper was determined. The tailing from the round-table feed contained 0.141 gram of soluble copper per liter of solution. The tailing from the round-table tailing contained 0.084 gram of soluble copper per liter of solution.

Disintegration of Flotation Froth.—In order to break up the froth in the flotation concentrate a saucer-shaped disk 3 ft. in diameter was constructed. This was revolved at a speed of 300 r.p.m. inside a circular tank. The concentrate pulp was fed on to the disk at the center and was thrown out against the sides of the tank. The impact broke up the froth to a great extent. No test was made on the disintegrator.

It was noted early in the experimental work that an elevator made a good froth breaker, the impact of the pulp discharge at the head of the elevator serving to break up the bubbles, to a large extent. A test made at the experimental Callow flotation plant showed that the rough concentrate pulp was reduced 75% in bulk in passing through an elevator. As to apparatus, it was concluded that the Minerals Separation was best adapted to the work at Anaconda.

CHAPTER VIII

THE USE OF PETROGRAPHY IN FLOTATION

A feature in the study of flotation that has rarely been touched upon, at least in the literature of the process, is the intensive examination of the ore to be treated. An interesting article on that phase of the subject was published in the *Engineering and Mining Journal*, May 6, 1917, by Donald G. Campbell. It is reproduced here as an exceedingly valuable example of the pertinacy of side lights on a major theme.

Petrography furnishes an exceedingly simple yet powerful method of attacking problems of ore deposition and through the light it shed on the nature, size and relationships of the mineral particles composing the rock, can help solve many of the problems that confront the metallurgist. Very often a comparatively trifling expenditure of time and expense will answer with accuracy such questions on the suitability of flotation, table concentration and the probable efficiency of separation of various sulphides in complex ores as would otherwise require extended laboratory or large-scale tests to determine. The merits of petrographic investigation need no argument to those dealing with large and many-sided problems of ore deposits, but it is too often ignored or overlooked as a collateral aid in small-scale work and preliminary investigations.

A series of laboratory tests was recently made to determine the suitability for flotation treatment of a refractory silver and gold ore in a Western mine. A stamp mill with vanner concentration had been erected years ago to treat this ore, but with indifferent results, about 50% being the best extraction obtained.

In the course of this work certain variations from expected results were encountered; the distribution of values as determined by screen analysis was inverted from what would have been usually expected, and as a result, while the greater part of the silver values (about 70%) responded readily to flotation when ground to the usual mesh, the remainder was found to be very resistant.

To attack the problem from a new angle, a petrographic study was made of the ore, by which the sequence of mineralization was clearly indicated. This study also furnished definite information on the dissemination and fineness of the individual mineral particles as affecting the degree of fineness necessary to liberate the sulphide mineral from the

gangue, and on the probable manner in which the barren gangue will crush as compared with the mineralized portion. The results so predicted were borne out and explained some of the difficulties encountered in the laboratory.

The ore is from an Idaho mine and is fairly typical of a number of Western silver deposits. The vein matter is largely quartz and in hand samples shows partly decomposed feldspar. The valuable mineral occurs mainly as fine dark banded disseminations along fracture planes running through the gangue. Frequent grains of chalcopyrite and pyrite ranging up to 2 or 3 mm. in diameter occur. These are surrounded by the more finely divided "sooty" looking mineralization which contains most of the values and form a "black quartz." This mineral aggregate is composed of various sulphides and traces of antimonides of silver, together with telluride, probably sylvanite and possibly also calaverite, the whole complex being such as was usually grouped together by the old miner of past decades as "sulphurets" and regarded by him with despair as far as successful mill recovery was concerned.

In thin section the material showed a groundmass composed largely of quartz individuals ranging in general from 0.5 to 2 mm. in diameter. Mixed with these were many orthoclase feldspars of about the same size. The latter were usually more or less corroded and decomposed. Minor accounts of accessory and also secondary minerals such as sericite, kaolin and calcite were present. Strain phenomena were shown by these primary minerals, indicating the dynamic agencies which led to the fracturing of the rock. These alterations tend to greatly diminish the original strength of the rock and decrease the difficulty of crushing. Close to the numerous lines of fracture occur the sulphides; and here many of the quartz grains show a zonal growth due to infiltration and deposition of the silica, with which the fractures were subsequently rehealed. It was indicated that the vein was derived from a very acidic aplite dike, with feldspar partly decomposed, greatly fractured by dynamic movement and mineralized in successive stages; and the whole mass was finally silicified and cemented together by the infiltrations introducing a part, at least, of the valuable minerals.

The sulphide minerals were seen to be grains of which a small proportion ranged up to 3 or 4 mm. diameter and lying alongside or in close proximity to the numerous fine fractures which formed the ore-carrying channels, and to consist largely of pyrite and chalcopyrite nuclei, more or less completely surrounded and replaced by argentite. Cutting into both the pyrite and argentite was another mineral, recognized as sylvanite. Minute amounts of other minerals were not identified, but were thought to be mixtures of the above with stephanite and possibly calaverite, among others.

Sulphides and Sulphide Minerals.—Beside these larger particles of sulphide, which formed perhaps less than half of the sulphide minerals,

were a great number of exceedingly minute grains of sulphide, seldom more than 0.1 mm. in diameter and usually much smaller. These lay either within the tiny channels of ore mineralization or closely adjacent, in a matrix of equally small quartz particles. These quartz grains were tightly interlocked with one another and with the surrounding groundmass of "barren" quartz by a zonal growth or later crystal development. This portion contained almost no alteration material and formed a cement-like mass, almost amorphous and of flinty texture, which would tend to lock up the inclosed sulphide particles very firmly. That this was in fact the case was well brought out by subsequent work in the laboratory.

In the microphotographs taken at 70 diameters' magnification, partly by reflected light, there is shown a large sulphide grain. The central nucleus is of lighter color (pyrite) while a darker growth surrounding it is mostly argentite, the whole in a groundmass of quartz. There is a tendency to crystalline outline shown by the argentite replacement, also by the minute ore channels or feeders. These larger particles would be easily recovered by table concentration to form a high-grade product, but with a low extraction—since these large grains represent probably less than 50% of the total sulphide content.

In material containing fine particles embedded in closely interlocked quartz groundmass, not only is fine crushing required to free the sulphides, but it would also offer much more resistance to crushing than the barren gangue.

A sample of the ore was crushed and a series of tests run under the following conditions: Screen analysis, + 80 mesh, 10%; 80 to 120 mesh, 25%; 120 mesh, 65%; oil, mixtures, 2 to 5 lb. per ton of ore; acid, sulphuric, about 5 lb. per ton of ore; pulp dilution, 1:5; temperature, normal; length of agitation, 20 to 30 min.; weight of ore charge, 500 grams; type of machine, mechanically agitated.

RESULTS OF FLOTATION TESTS

Run No.	Oil	Heads	Tails	Conc. os. Ag	Silver extraction
6.....	350 ¹	88.0	24.0	970.0	73%
8.....	1,580	88.0	27.3	831.0	69%
9 ac. sludge.....		88.0	24.2	774.2	72%
10 calol, E, 350.....		88.0	27.5	616.8	69%
16 350, mixture.....		88.0	24.0	759.0	73%

¹ Numbers refer to Pensacola oils.

Neutral and alkaline solutions were also employed but with less satisfactory results, the kaolin colloid producing a heavy, dirty froth. Oils ranging from mixtures of heavy pine oil, wood creosote, calol

mixtures, as well as gas and coal-tar oil were also tried. A mixture of pine oil, with a small amount of wood creosote, gave perhaps the most consistent results, although the same mixture plus coal-tar or gas-tar oil worked very well. In practice a mixture of from two to four parts of tar oil and one part of pine oil or wood creosote may be found efficient.

Another portion of the ore was then pulverized, sampled carefully and a screen analysis made to determine how the values were divided between the different screen sizes. In such ores it is usually assumed that the valuable content finely disseminated limes readily and becomes so finely divided that it is impossible to separate by gravity concentration. However, in this case, it is indicated by the petrographic investigation that a considerable portion of the valuable content is firmly locked in the fine-grained silica which fills the fractures through which the mineral itself is distributed, and it was inferred that this combination crushes much less readily than does the granular quartz comprising the greater part of the gangue. This being true, much finer grinding is necessary, to free the valuable mineral so inclosed. That such actually is the case is strikingly confirmed by the analysis which follow. The first screen analysis showed such a complete reversal of the commonly expected distribution of metallic content that it was checked by a second test, which checked the first very closely and confirmed the prediction made as a result of the petrographic work as to the difficulty of releasing the fine metallics.

SCREEN ANALYSIS OF HEADS

Gold, 0.77 os. per ton; Silver 98.0 os. per ton									
Test No. 1					Test No. 2				
Approximate screen aperture	Mesh	%	Ag. os. per ton	Prop. Ag. os. ¹	%	Au. os. per ton	Prop. Au. os. ¹	Ag. os. per ton	Prop. Ag. os. per ton
0.16 mm.....	+80	10.0	197.8	19.8	8.4	1.10	0.09	187.5	15.8
0.14 mm.....	80 to 120	25.5	105.2	26.9	29.0	1.00	0.29	116.0	33.7
0.12 mm.....	-120	64.5	80.0	51.6	62.6	0.64	0.40	80.0	50.1
		100.0		98.3	100.0		0.78		99.6

¹ Proportion of the total value per ton remaining in the respective sizes of product, expressed in ounces per ton; total should check the value obtained in the head assay.

The two screen analyses check closely and show to what a marked extent the coarse particles hold the metallics locked up. That the metallics were not present in coarse concentrates was indicated later by the fact that panning yielded only a negligible percentage of concentrates. Several runs were next made to again check the efficiency of extraction, and in run No. 21 the concentrates were recleaned to a higher grade.

No.	Run	Heads	Tailings	Concentrates	Extraction
19	Gold.....	0.77	0.10	3.20	87%
	Silver.....	98.00	26.00	673.00	74%
21	Gold.....	0.77	0.10	6.00	87%
	Silver.....	98.00	22.00	1,255.00	78%

An analysis was made of the recleaned flotation concentrates from run No. 21 to determine its composition and to deduce therefrom what grade of concentrates it should be possible to make in practice. The analysis was: Au, 6 oz.; Ag, 1255 oz.; Cu, 3.36%; insol. 47.3% Fe, 19.5%; Te, 0.86%; S, 12.9%.

From this it is evident that if it were possible to make concentrates absolutely free from silica, the grade could be almost doubled. But as the silica can probably not be lowered below 20 to 25% without an undue increase in the grade of tailings, it is evident that from this class of ore the probable maximum content will be 9 to 10 oz. of gold and 1800 to 2000 oz. of silver.

In order to ascertain how much of the remaining silver would be amenable to flotation when more finely ground, the tailings remaining from several of the foregoing runs, including Nos. 19 and 21, were reground and again floated, with the following results:

	Orig. heads	Tailing	Conc.	Extraction	Total extract
Gold.....	0.77	0.55	1.75	93.5—	87.7
Silver.....	98.00	13.30	222.50	86.5—	

This is of course a middlings product which in practice would be returned to the circuit for recleaning.

In floating the reground tailings the concentrates formed much more slowly than with the original ore; the extraction was improved and the length of agitation diminished by sulphidizing with a small amount of hydrogen sulphide. The 13.3 oz. of silver still remaining in the tailings is too high to be regarded as very successful work, but it is believed that complete sliming of the tailings will recover a good part of the refractory mineral, and further investigation of the tailings was not considered necessary at the time.

Extended tests being made on silver ores of similar nature have encountered similar conditions, and final success has been attained only by complete sliming of the tailings and the use of sulphidization. Sliming, indeed, had been proved necessary in the treatment of this class of ore by cyanidation, and it was not to be expected that anything less would give the highest extraction, even in flotation.

The necessity of sulphidization is not so obvious. No trace of oxidation of the ore could be seen, and I incline more to the belief that

it is due to the fact that even the slimed particles of sulphide are mixed or coated with sufficient silica to overcome their flotation properties, except where the very buoyant influence of fresh sulphidization has been applied.

In a small mill this resistant portion of the tailings would be reground by being returned to the tube-mill circuit with a classifier of the Dorr or drag-belt type, but in a mill of, say, 200 tons daily capacity or greater, I believe that it would be better practice to send it to another small tube mill, to be slimed separately.

CHAPTER IX

THE APPLICATIONS OF FLOTATION

Among metallurgical processes flotation is apparently unique in that it is applied to a wide variety of ores. Many operators have expressed surprise at the extent of the field for flotation, but when one considers that it is merely a variation of concentration and that all possible kinds of ores are concentrated, it is not unusual, after all, that it should be applied to so many ores. Flotation really is not a complete metallurgical system, but only an incident or a step in the long process of extracting metal from its ore. Flotation does not extract metal from any ore, but simply separates the greater part of minerals from a large proportion of gangue, just as any other concentration process is designed to do. Flotation does it, on many ores, so much better and so much cheaper than gravity concentration that it is being widely adopted, and the method deserves all the attention it can get.

The process is so new that clearly defined systems of application have not yet been pointed out. At present it is merely an empirical matter involving an extraordinary amount of testing to see, first, whether flotation is adaptable to an ore, and second, if it is, how best to adapt it. No one process or system will apply to all possible ores, even of one metal, and flotation has up to this time been applied to copper, zinc, lead, molybdenum, gold, and silver, with possible applications in some other lines.

Some Fixed Principles.—There are, however, a few principles that can be considered as practically fixed. One of the most definite of these is that oxide ores, as such, are not floatable. As a rule it is impossible to get an oxide, *per se*, to float, although it is possible to change the surface film of an oxide particle into such a form that the particle will float. The reasons for this have been discussed in the chapter on theory, and from that it has been determined that the contact angle of an oxide particle is not such as to permit the necessary application of surface tension forces. Oxide particles seem to act with oils very much as

gangue particles do, refusing to form a film upon them of such character that the particle will be enabled to present a contact angle such as to enable it to float. Carbonate particles are also difficult to float, but sometimes it can be done. In general, however, it may be said that the carbonate particles follow the inclinations of the oxides and do not float unless a surface film of proper kind has been formed upon them.

The Character of Colloids.—Another definite fact that is fairly clearly recognized is that colloids are not favorable to flotation. This has also been discussed in the chapter on theory, the basic reasoning being that colloids are particles so finely divided that they approach the solution state, and it will be admitted that particles or molecules in solution are not conducive to commercial flotation. The dispersion is such that what are ordinarily called flotation tendencies have little effect upon them. It is not impossible, however, to practice a selective flocculation and sedimentation, a point that has already been touched upon in the chapter on testing ores. A thorough study of colloids is of great value to the student of flotation as well as to one who cares to understand thoroughly the basic principles of any wet milling process. The literature of colloids is not rich, in English, and perhaps the clearest available explanation of the subject is contained in the series of articles by E. E. Free.¹ A thorough study of this enlightening exposition will be of great benefit to any one interested in ore dressing, but it is too lengthy to be touched upon here. So far these are about the only points that have been definitely settled, although some others seems to be fairly apparent. It is true that sulphides are floatable, possibly the most easily floatable of all the minerals, and in general it may be said that all sulphides are susceptible to the flotation process, but even this must be somewhat modified, since with some of the sulphides better results are secured by some other process, and some complex sulphides do not even give satisfactory results through the application of any of the variations of flotation now known and practised.

Flotation to Produce Two or More Products.—In addition to the applications of flotation to ores from which one product is to be made, there are other ores in which several products may or must be made. Such ores constitute a distinct problem and they must be worked out, each on its own merits. Thus, when an ore

¹ "Colloids in Ore Dressing," *Eng. and Min. Journal*, Feb. 5, et seq., 1916.

containing zinc and lead is to be treated, and this intermingling of the zinc and lead constituents is intimate, as it usually is in such ores, a different scheme of treatment must be inaugurated. Some of the most important uses of flotation are in just such cases, an example of which is found in the Coeur d'Alene district of Idaho. Here zinc, lead, and silver occur in the same ores, sometimes together with copper. All of these metals are to be recovered and as nearly separate as it is possible to do so. This is being done successfully, the ways and means for which will be detailed.

To begin with the simplest problem would be the most logical process, therefore it would be well to consider the recovery of a single sulphide, take copper for example, from ordinary ores. The first thing that should be recognized is that it is almost impossible to secure anywhere near complete extraction and high-grade concentrates at the same time and through the same operation. The procedure, then, is to make a first concentration in which the greatest possible percentage will be recovered from the ore without regard to the grade of the concentrate secured. The second operation, then, is to work on the low-grade concentrate secured, and by a repetition of processes to segregate the valuable portions into still smaller bulk, leaving a low-grade middling for further treatment. This middling is usually either reground or returned to the secondary system for retreatment. The two processes are in general known as "roughing," that is the first process of extracting as much of the mineral as possible from the ore bulk, and "cleaning," that is the producing of high-grade concentrate from the low-grade material first extracted from the ore bulk.

Flotation a Principal or Secondary Process.—Before going into the actual flotation work, however, one has to consider, in a scheme of treatment, whether or not flotation is to be the principal treatment or whether it will be a secondary consideration. To say that flotation may be the sole process of metallurgy in a given case, is to make a statement entirely too broad for general use. Some ores, it is true, may be treated by flotation only, but it is reasonably certain that such ores are limited in number and that the statement can by no means be made a widely applicable rule. Ores that are coarsely crystalline in nature can best be treated by stage crushing and stage concentration, beginning with jigging. By this means a large amount of expensive grind-

ing and regrinding is avoided, and concentrates are recovered as quickly as possible. Of course, in such case, the finely divided part of the ore, or slimes, would probably be best treated by flotation, but with an ore of this character there is no reason for doing any more grinding than is just enough to liberate the minerals and allow them to be separated from the gangue. It may be said that this particular point is just the one that limits a reasonable application of flotation and other concentrating systems. The mineral particles must be liberated from the gangue by crushing and then separated by concentration. Therefore crushing should be carried only far enough to liberate these particles, and the concentration process then applied to separate one from the other should be appropriate to the size of the grain thus freed. If the grains are all coarse, then under ordinary circumstances jigging, table concentration, or any form of ordinary gravity concentration is the one that should be applied. If the grains are fine, then flotation may do the better work and at a lower cost. Of course it may be possible that in special instances, such as with very heavy gangues approximating the weight of the mineral itself, fine grinding and flotation will have to be resorted to in order to get satisfactory results, no matter what the size of the mineral grains.

One authority, referring to the discussion as to whether or not flotation should be used as an auxiliary, said that were flotation always considered as secondary instead of a primary process, a large portion of the benefit of its economy of operation would be lost, and while it is true that some ores would justify coarse crushing, jigging and table work previous to flotation, a large number of ores, especially the lower-grade porphyry copper ores and the closely included lead ores, will show a distinct benefit by making flotation the primary treatment. Where initial crushing is fine enough for flotation, and where such crushing can be obtained entirely by a crusher followed by a ball mill, an extremely simple and effective flow sheet can be formed. In most cases where this is done, the tailings from flotation are run over tables of the Wilfley or Deister type. As previous to this tabling practically all the fine material that can be saved has already been removed by flotation, the table feed requires no preparation by sizing or classification and a large tonnage can be run over each machine. Even on ores where the mineral

occurs coarse, we are beginning to believe that the best method of treatment will be to crush to six, eight, or ten mesh, rough on tables of the ordinary roughing type, recrusher these rougher tailings and send them direct to flotation. Flotation treatment has proved itself to be not only extremely efficient but permits a simple flow sheet, cutting out a large number of elevators, trommels, and other kindred annoyances.

Floating a Selected Portion of the Ore.—While there is much of logic in what this authority points out, it is somewhat difficult to accustom oneself to the idea of concentrating on tables a flotation tailing. It would ordinarily seem more reasonable, as I have already pointed out in a previous paragraph, to crush the ore only to that fineness required to liberate the mineral grains. Now, if a large quantity of mineral grains can be liberated at, say 10 mesh, and a certain percentage of that mineral can be recovered by jigging or tabling, then it would seem unreasonable to crush that same material to 60 or 80 mesh in order to treat it by flotation. It is not as if the same bulk had to be treated by flotation anyway, since it is not always impossible to separate by jigging, tabling, and other well-known milling methods, first a high-grade concentrate to be regarded as a finished product, a middling product containing practically all the remaining value of the ore, and a tailing that can be discarded. If we subtract from the original ore bulk, then, the finished product that is saved and the tailing that is discarded, we have left only the middling to be reground and treated by flotation, thereby reducing materially the tonnage to be handled by that process. All in all, it seems a very cheap and reasonable way of going at it. However, since flotation has seemed to reverse practically all of our theories, many of which we formerly held dear, I am not going to make the mistake of advocating too strenuously any one procedure, since I am of the opinion that flotation is by no means in its finished form at the present time, and there may be many improvements and developments that will change all the ideas that we have formed.

Another authority, speaking of this same subject, points out that it is usually not advisable to draw a line of demarcation between sand and slime, reserving only the latter for flotation. In this case, the reference to slime does not mean colloids but only the very fine portion of the ore which may be granular or not as the case may be. As we have already seen, flotation

does not apply to colloids, but may apply to very fine material when it is flocculated. A system that has often resulted in improved mechanical efficiency, is to separate sand for table concentration, making a clean, high-grade concentrate without trying to clean absolutely the tailing, and then to regrind the tailing and add it to the slime, the whole being then subjected to flotation. This authority, however, points out that this is especially true for gold and silver work. In general, when the concentration ratio is high, that is when there is only a small portion of mineral in comparison to a very large proportion of gangue, authorities seem to believe that flotation would be the best sole process to be applied. In such cases gravity concentration is considered an auxiliary. When the ratio of concentration is low, it is considered better practice to begin with gravity concentration and finish up with flotation. This is practically as I have outlined it in a previous paragraph, and some ores are such that they will be well handled by this process. In cases where new and modern concentration mills have been constructed based on gravity practice, it would be unwise to waste all the money involved in that work, and the point of wisdom is to use flotation as an auxiliary to the gravity concentration scheme.

As an example of ores in which the ratio of concentration is very high, we can point to the porphyry coppers, such as Inspiration, Miami, Utah Copper, Ray Consolidated, and the others of like type. Here the percentage of copper is very small and is highly disseminated through a large proportion of gangue. Here, so the copper authorities point out, it is best to apply flotation at once, since the mineral particles are so small that the ore has to be ground very fine to liberate them anyhow, and that their handling by flotation alone requires a much cheaper and simpler installation. If it should develop, however, and the scheme is by no means impossible, that the finely ground porphyry copper ores can be treated primarily by selective settling in some such machine as a Dorr thickener, it would be better to eliminate as tailing the largest possible percentage of such material, reserving only the part containing mineral for flotation concentration. The question of selecting settling has already been treated by O. C. Ralston,¹ who points out his practical experiments along that line and demonstrates

¹ "Control of Ore Slimes," *Eng. and Min. Journal*, Apr. 29, 1916.

the possibility of commercial applications of the system. They have not been successfully accomplished yet, but are by no means impossible.

It seems that flotation processes are limited on one hand by the size of the particle and it is useless to treat a pulp by the flotation process unless all of it has been ground to such a point that the particle will float under the best obtainable conditions. There would be little use, for example, in attempting to float a $\frac{1}{4}$ -in. mesh product, and while it is likely that no one would be foolish enough to make the attempt, the line of limitation between a floatable particle and a non-floatable one is not clearly enough defined in the mind of the ordinary operator. The size of the particle is one of the important influencing conditions, and all operators should recognize that. If it is possible to make a good product by gravity concentration on coarse material, it should certainly be done, since the expense of fine grinding is then avoided. But where the mineral is extremely finely divided and disseminated in the ore, there is no particular advantage in performing stage concentration because no high-grade product can be obtained on coarse grains.

The procedure already referred to as being applied in the Coeur d'Alene, where zinc, lead, and silver are to be obtained by concentration, may now be investigated. Most of the mills in that district have been using gravity concentration for a number of years. The process is started with jigging, and then proceeds to coarse-table and fine-table work, finally ending up with flotation on the finer products. It is to be remembered, however, that but few of these mills are new, and many of them contain a heterogeneous collection of concentration machinery that represents a gradual development from the first steps of the process up to the most modern improvements. One of the new mills in this district is that of the National Copper Mining Co., and in this plant, which will be treated in detail in another chapter, the ore is subjected to crushing, screening, classification, and table concentration, and also regrinding, before going to the flotation machine. A system of primary roughing and final cleaning is carried out.

The Coeur d'Alene District.—At the older mills of the district it has been difficult to secure permission to publish data, and for that reason no definite statements can be made at the present time. One interesting fact, however, that has been made

public before is the installation of a Macquisten plant at the Morning mill of the Federal Mining and Smelting Co.

In general the ores of the Coeur d'Alene are such as I described¹ in an article on this subject. Gold and silver have been found in native form but do not usually exist that way. Among the minerals occurring in that country, galena should be mentioned first since it is the most important, being the source of a large production of lead and the medium that carries a great part of the silver. It is probable that galena is the most widely distributed mineral in the district with the possible exception of pyrite. The sulphide of antimony (stibnite) is not rare, and is common in the Wardner district. Chalcocite is found occasionally, but is rare. Sphalerite, a well-known zinc mineral, is rather widely distributed, but in quantity not so great as might be expected in view of the galena quantity. Siderite, the carbonate of iron, is the most abundant gangue mineral of the Coeur d'Alene silver-lead deposits and is one of the most important features that has to do with the metallurgy of the district. The minerals occurring there form a hard combination to handle, and flotation has been a great assistance to the profit-making endeavors of the metallurgists concerned.

At the Gold Hunter mill a complete system of stage crushing and concentrating is in force, the concentration beginning with jigs and progressing through tables down to finally the roughing and cleaning flotation machines. At this mill the Callow pneumatic system is in use, as it is in a great many other plants in the district.

At other mills of the district, the feed is all crushed to pass 100 mesh, although this is not usual. At one mill, material passing 60 mesh is treated by flotation, the tailings being then re-concentrated on gravity tables. Some authorities claim this to be rather poor practice, since none of the galena products between 60 and 100 mesh will float, and must be caught on the tables. In this case, it would probably be better practice to treat the + 100-mesh material on tables first, making a clean tailing and low-grade middling, the latter of which could be reground and then treated successfully by flotation. The question of oils in the Coeur d'Alene is not yet settled, but the practice is in line with that of other progressive districts in which

¹ "Metallurgy in the Coeur d'Alene," *Eng. and Min. Journal*, Nov. 20, 1915.

a larger quantity of an oil to be used for oiling the mineral particles is combined with a much smaller quantity of a frothing oil.

In the Globe-Miami district, of Arizona, several plants are using flotation, and at least one of them has been built with an idea to incorporating flotation in the principal flow sheet. At the Inspiration mill, for instance, ore is crushed in ball mills, into which the flotation oil and mill water is also added. The material is then classified and the slime thickened in Dorr machines, the underflow from which runs to flotation. The sand portion is also sent to flotation, but through a different set of cells which are operated in a slightly different way. At this mill there are several different kinds of machines, including the Callow, the Minerals Separation and a special type of machine developed at the works. At the Old Dominion mill, the ore is somewhat different from that treated at the Miami and Inspiration. In this case jig concentrates are made and Minerals Separation machines are used for treating the slimes. Flotation here produces a sort of an end product, and is only a small portion of the entire mill concentrate. At the Miami mill, Callow cells are used for final treatment of slimes, since this mill was not designed primarily to use flotation.

Floating Precious-Metal Ores.—While flotation has not been extensively applied to silver-gold ores, there is no reason why gold-bearing sulphides or gold-bearing silver sulphides, or silver sulphides alone, should not be treated, at least in part by flotation. A noted authority on cyaniding has announced his belief that the treatment of ordinary silver-gold ores will be, in the future, a combination of cyaniding and flotation. By this system a stage-grinding series is carried out, no doubt as has been done heretofore, beginning in crushers and continuing with either stamps, rolls, chilean mills or some such intermediate grinding machines, and finishing up with tube-mill grinding. The products from the tube mills will then probably be classified, a selected fine portion going to flotation, the sand portion being concentrated on tables and then leached by cyanide. This would probably make a very cheap process to operate and a rather inexpensive one to install. The alternative, as has already been pointed out, is to regrind the sand after table concentration and send it, too, to flotation. In many cases this might be a more satisfactory process, particu-

larly if the table product is divided into high-grade concentrate, low-grade middling, and barren tailing. In this case the latter could be discarded altogether and only the middling reground and treated by flotation, making a very simple flow sheet to maintain.

In this connection it is interesting to note that flotation has been applied to the silver-sulphide ores of the Santa Gertrudis mine, at Pachuca, Mexico. There was said to have been no difficulty in successfully floating the mineral. Again, flotation has been successfully practised in some of the mills of Cobalt, Ontario, where silver occurs in very complex form. Only recently, however, it was announced that flotation, by the Callow system, had been discontinued at the Nippissing mill because of the difficulty of floating the tailings from cyaniding. An explanation will be extremely interesting.

The procedure to be followed in installing flotation is, of course, one that has not been definitely settled for all cases and one that requires considerable study to work out in the proper way. The natural development of the process is in pointing out such facts, from time to time, as will add to the possibility of forecasting results that will follow certain systems. At present, however, a great deal of study and a long, careful persistent system of experimentation is necessary in every application.

CHAPTER X

EXAMPLES OF FLOTATION PRACTICE

Since the first days of flotation the process has appeared particularly attractive to Australian operators who have given a great deal of attention to its development. It would not be too much to say that in the early days the Australians were the only ones who seemed to place much serious importance upon this process and many important facts were discovered there. Naturally, in the incubating days, the flotation process proved to be a continual source of trouble and exasperation to its operators, and the Australian metallurgists deserve a great deal of credit for the successes they have extracted from a system that did not appeal strongly to operators in other parts of the world. Their perspicacity has been fully justified by recent developments of the art and the spread of flotation to all corners of the universe.

In the early days those who are familiar with Australian metallurgical development agree that flotation with and without oil occasioned more hard labor and real desperation than anything that has appeared in recent years. For this reason it is of interest to note some of the successful plants that have brought Australia to the front in flotation practice.

The Central Mill, Broken Hill.—A paper by James Hebbard¹ describes the operation of the process at one of the large mills in Australia. He points out that while concentrating the galena in the lead ore produced from the Central mine, a valuable by-product was obtained in the form of slime assaying 18% lead, 20% zinc, and 16 oz. of silver per ton. This came from an ore in which the quartz and rhodonite were the chief gangue minerals. In the course of ordinary operations it had long been observed that a froth was formed containing high metallic values, in silver and lead particularly, whenever conditions were favorable, as for instance, where the rotation of

¹ "Flotation at the Central Mine, Broken Hill," *Proceedings of the Australasian Institute of Mining Engineers*, Nov. 10, 1913.

trommels, or the splash of the elevators or the motion of jig plungers produced a violent agitation of the mill water containing slime. Early in 1901 a series of experiments was carried out for the purpose of reproducing and accentuating the conditions responsible for this valuable float concentrate. Experiments and tests, extending over several months, were made on slime of various degrees of fineness. Among the appliances tried was a series of funnel-shaped vessels with the small ends downward, each fitted with an overflow lip. The bottom end of the funnel or cone-shaped vessel was fitted with a cap or plug discharging the contents into the next vessel of the series and so on. To each vessel was attached, near the bottom, a water pipe as well as a pipe carrying compressed air. The object of the water was to provide an upright current for the contents of the vessel, while the object of the compressed air was to produce agitation and enhance the agitation effect of the upcurrent of water with the expectation of reproducing the conditions causing the float or metallic froth. These experiments were on the lines of a spitzkasten with a strong upward current to produce an agitation on the slime water, assisted by jets of compressed air. It was thus early recognized that the bubbles of froth noted in the wet concentration operation were due to the aeration produced by violent agitation, resulting from mechanical implements moving rapidly in water. In these experiments a metallic froth or scum could be produced and recovered assaying 36 oz. silver, 30% lead, and 22% zinc.

Early Flotation Experiments.—Early in 1903 an exhaustive series of laboratory tests was made on the lead by-products by flotation methods, using heated sulphuric acid and salt-cake solution. These tests yielded some success on materials specially prepared, that is on grainy material from which both the coarse and the fine had been eliminated, leaving it evenly sized. Certain classes of the material produced by the mills contained such a large proportion of carbonates—such as carbonate of manganese, lime, and lead—that no flotation could be secured except by a prohibitive consumption of acids. These tests were carefully made but the best work done in the laboratory was not equal to that being secured on a commercial scale in the existing magnetic plants. The testing of flotation methods were conducted in pans or vessels worked on the principles of spitzkasten, following the lines of Potter and Delprat.

The first apparatus was constructed so that the liquids could be raised in temperature by the application of direct heat underneath the pan, but in the later types the temperature of the liquor in the pan or spitzkasten was maintained by the injection of live steam into the storage tank. In none of these tests was agitation employed, the material to be treated being fed practically dry on the surface of the liquor in a regular stream and heated liquor added through a pipe discharging near the bottom of the vessel. A still surface was imperative in this operation and it was equally evident that the operation depended largely for success upon the surface tension of the liquor after the gas evolved by the action of the acid on the mineral carbonates had raised the particles. This surface tension was increased by the density attained in the one case by the salts formed from the mineral and gangue through the action of sulphuric acid and the other by the addition of salt-cake. In all of these experiments the liquor was returned by the ordinary type of air-lift, using compressed air at about 70 lb. pressure.

Experiments definitely demonstrated that there was a limit to the size of the particle that could be buoyed up. Also that any material below a certain size, no matter what its character—whether gangue or mineral—floated because of the density of the solution. And in addition, that if the finer particles of gangue were not eliminated before treatment they would be floated with the minerals and lower its value in metals to such an extent as to make it unmarketable. Slime, whether existing as a by-product of the ordinary wet-mill concentration or subsequently produced in preparing tailing for treatment, forms such a large part of the material available for treatment that any process that failed in this direction was too limited in its scope to be of value to the Central mine.

As far as any process up to date was concerned slime had to be regarded as of no value except in so far as it was available for smelting. The Broken Hill Proprietary Co. had used a considerable quantity in this way and had discovered that roasting or sintering this slime in open heaps gave a fair product valuable for smelting, a good deal of the sulphur and a fair proportion of the zinc having been driven out. The Sulphide Corporation also sought to make its slime available by this means, but it was proved that the losses of metal were too great to justify the method.

These experiments were abandoned and tests were made then on the Cattermole process. This process seemed applicable and a plant was constructed to take advantage of its principles. The plant ran a few months on tailing and slime from various sources and during this time the frothing method was generally ousting the granulation process, until finally the superiority of the froth method was clearly demonstrated. The froth method, thus, was a direct outgrowth from the granulation process at the Central mine. The Wilfley tables of the original plant were then dismantled to make room for the sand boxes and the granulation process gave way to flotation with simple spitzkasten.

Introduction of the Froth Method.—Finally Minerals Separation, owners of the froth patent, purchased the tailing dump on the Central mine and by arrangement a plant was designed and erected by the Sulphide Corporation for their treatment. This plant was designed on previous experience for the treatment of 2,000 tons per week. It was finished at a cost of £26,000 complete. The efficiency of grinding pans proved so great with the new design of positive pan that the plant was easily able to handle 5,000 tons per week. This plant was responsible for the treatment of 7,999 tons of tailing and the production of 242,462 tons of concentrate up to the time it was shut down in June, 1911, upon exhaustion of the dump.

In connection with the Minerals Separation plant, it is important to note that the fact of the circuit being no longer acid, but neutral, has been taken advantage of, inasmuch as there is only one circuit through from grinding pans, agitators, and spitzkasten. The original granulation plant, being designed for acid circuits, was equipped originally with wood throughout where liquor circulated, and with dry-crushing ball mills for the same reason. Later, wet crushing was adopted, but with a fresh-water circuit, kept carefully separate from the acid circuit in which the actual separation took place. The successful development of the flotation process, however, has enabled both crushing and separation to be conducted in one and the same circuit and has thus greatly simplified operations.

The liquor that was circulated through the Minerals Separation plant was approximately 25,000 gal. per hour. During the course of operation, therefore, nearly 3,000,000 tons has passed through the twelve iron grinding pans of this plant without

detrimental effect. No stronger evidence could be produced as to the freedom of the circuit liquor from acidity. The maintenance charges on these iron pans are no heavier than corresponding charges on exactly similar grinding pans in the lead mill crusher section, where fresh water only is used.

Selection of the Oily Agents.—Following exhaustive experiments in the laboratory various media have, from time to time, been used for long periods on the commercial scale, both in substitution for and in combination with oleic acid. Chief among such media are amyl alcohol, resin oil, camphor, pine oil, and eucalyptus, with all of which ingredients good work has been accomplished. It was in the milling plant on the Central mine that eucalyptus oil was first used on a commercial scale for concentration by flotation.

This satisfactory stage having been reached, attention could be turned to improvements in methods of handling, and on suggestions from the owners of the patents it was found that by connecting the bottom of each spitzkasten with the bottom of the next mixer in series, all of the spitzkastens could be kept on one floor, thus improving the supervision of the work. An experimental plant of this nature was erected at the end of No. 2 section in September, 1910, and, proving highly successful, the system was altered to that method of working. Experience shows that for ideal work the feed material should all pass through 40 mesh, but it is impossible to secure this condition of grinding at all times in the mill as designed. Moreover, although the Sulphide Corporation was aware that improvements in character of plant and methods of operation were easily possible, their attention, by force of circumstances, had to be turned sedulously to increasing the production with the appliances at hand.

Grinding for the Flotation Process.—Illustrating what is meant by ideal grinding, the records of the average assays of residues show continuously that where the average in zinc is from 2 to $2\frac{1}{2}\%$, that proportion remaining on 40 mesh will assay from 3 to 4% zinc. As illustrating the character of feed, and proving that the process is capable of handling successfully the very finest material, sizing analyses by commercial screens of the feed to the zinc section, zinc concentrate as shipped, and de-leading plant lead concentrate as shipped, are given in the accompanying table.

Feed to zinc section			Zinc concentrate			De-leading lead		
Through	On	%	Through	On	%	Through	On	%
	40	11.2		40	1.5		40	1.9
40	60	21.4	40	60	16.9	40	60	7.5
60	80	19.4	60	80	21.1	60	80	15.8
80	130	15.6	80	130	21.7	80	130	18.8
130	180	7.3	130	180	7.4	130	180	8.5
180	25.0		180		31.4	180		47.5

For comparison with the work at the old mill, the following summary of results achieved by the existing plant will be of interest. This table summarizes work done on a commercial scale at the Central Mill over a period of 12 months, ending December 28, 1912, and demonstrates conclusively the vast improvement in concentration practice made possible by the adoption of the flotation process.

SUMMARY

	Assay value				Recoveries		
	Prop. %	Ag. os.	Pb, %	Zn, %	Ag, %	Pb, %	Zn, %
Lead concentrate ex lead section.....	16.0	33.1	67.0	6.7	44.7	72.4	6.1
Lead concentrate ex de-lead plant.....	1.4	50.9	62.5	12.5	5.9	5.8	1.0
Total lead concentrate.....	17.4	34.5	66.7	7.1	50.6	78.2	7.1
Zinc concentrate.....	32.7	16.1	8.1	46.4	42.1	15.9	84.7
Total concentrate.....	50.1				92.7	94.1	91.8

The Silver Peak Mill.—An article by H. Hardy Smith¹ describes another successful application of flotation in Australia. This small mill was erected at the Silver Peak mine for the purpose of treating the second-grade ores of the Yerranderie silver-lead mining field, New South Wales, Australia. Owing to the distance of the field from smelting works and the high cost of transport, over bad roads by horse teams, only the richest ores could be shipped at a profit. The question of smelting locally was studied, but owing to the highly siliceous nature of the "seconds" (about 70% SiO₂) and the difficulty of procuring suitable fluxes, the idea was abandoned. The question of concentration was then considered, the problem proving to be a difficult one. The metallic minerals in the ore were, in order

¹ *Eng. and Min. Journal*, Dec. 11, 1915.

of importance: Galena, pyrite, marcasite, arsenical pyrite, blende, chalcopyrite, and small quantities of secondary minerals from all the foregoing.

All these minerals carried silver to a greater or lesser extent, but the chief silver-bearing constituents were the galena, and the iron sulphides, especially the marcasite. The galena occurred in two crystalline forms, one coarse and the other fine, the latter having a fracture similar to hard cast iron. This was rich in silver. Being extremely soft and friable, it slimed badly on crushing. Some of the ore contained it in such a fine state of dissemination that it appeared as a dark streak through



FIG. 32.—The Silver Peak Mill, Yenanderie, New South Wales, Australia.

the quartz, necessitating very fine grinding to free the values. The gangue was chiefly a hard glassy quartz with small quantities of felsitic country rock. From this description of the ore it will be readily understood that concentrating it was not simple.

Experiments were first made with gravity concentration, but the recoveries, especially of the silver, were low. The coarsely crystalline galena presented no difficulties, being amenable to jigging and table concentration; but this type was relatively low in silver, and the fine-grained variety, carrying the greater part of the silver, slimed so badly that no system of gravity concentration could give satisfactory recoveries. Flotation concentration was therefore tried, and as good results were reported from small-scale tests made by Minerals Separation, a plant incorporating that process was decided upon.

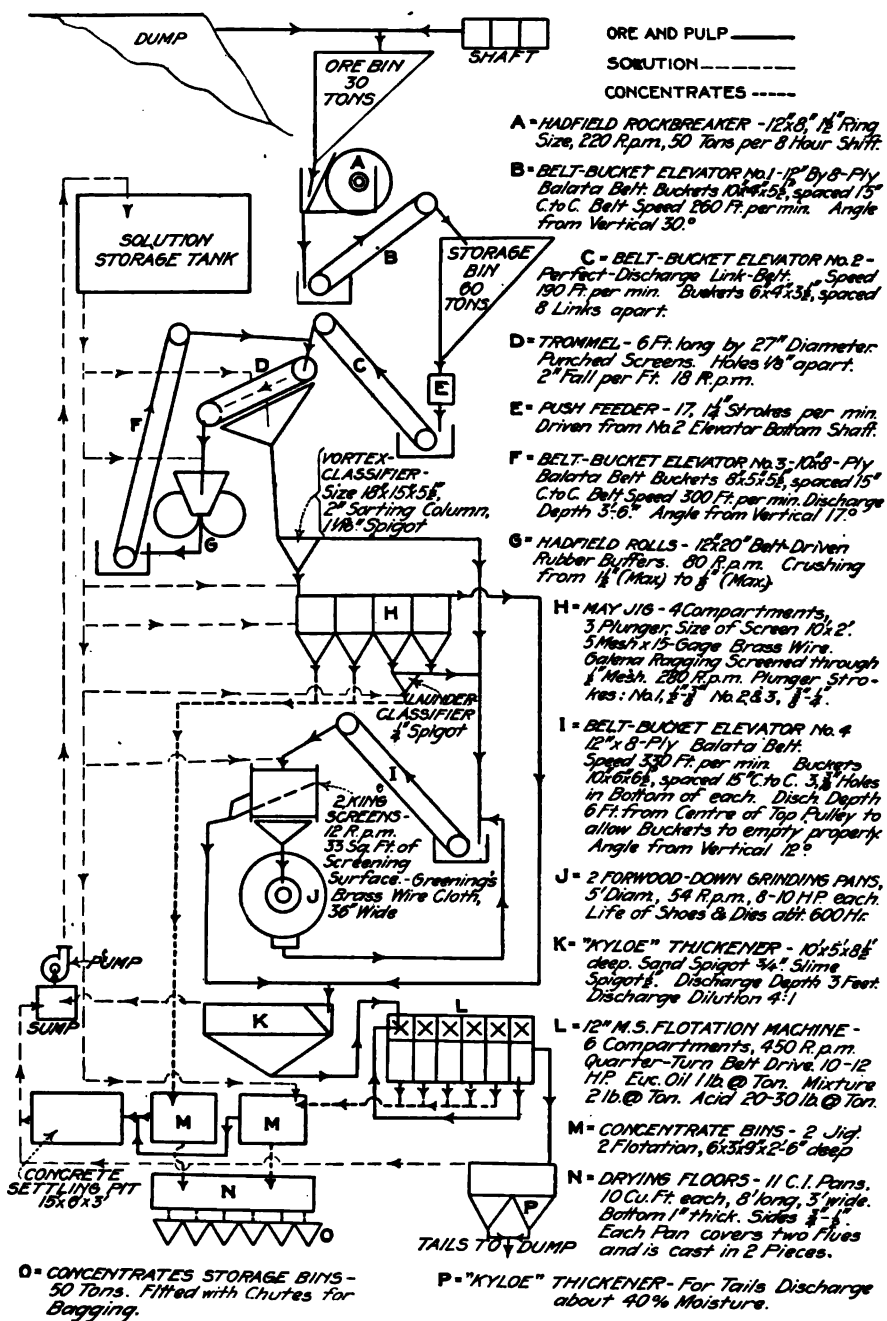


FIG. 33.—Flow sheet of the Silver Peak mill.

The average assay of the ore to be treated was about 30 oz. silver, 10% lead and 1 to 2 dwt. gold per ton. Because of the excessive transportation charges, the aim was to recover as much as possible of these metals in the form of high-grade concentrates. As the orebodies on the field were narrow and limited in extent, the company desired a mill of small capacity (about 50 tons per day) and small capital cost, which proviso precluded anything of an elaborate nature in the design. Several thousand tons of "seconds" had been mined during the opening up of the high-grade shoots of ore and lay awaiting treatment. The plan was to get the mill started and "tuned up" on this ore, supplementing it with ore from current development and, on results proving satisfactory, to mine the second-grade ore on an extensive scale. The flow sheet of the mill as erected is shown in the drawing, Fig. 33.

The thickener before the flotation machine was of the inclined-baffleboard type already described.¹ They are cheap to build and simple and efficient in operation. The clear overflow from the thickener ran to the pump sump. The thickened underflow ran to a 6-compartment, 12-in. standard Minerals Separation machine. The tailings from the Minerals Separation machine ran to another thickener, the clear overflow from which ran to the pump sump, and the thick overflow into trucks and thence to the dump.

Handling the Concentrates.—The flotation concentrates ran to a pair of bins, one being filled while the other was settling and being emptied. The turbid overflow from these bins ran to a concrete-lined pit where it was allowed to settle completely, the clear overflow from the pit running to the pump sump. This pit was cleaned out at intervals. After siphoning off the excess water from a concentrates bin, the sloppy concentrates were shoveled onto cast-iron drying floors and when dried sufficiently, into bins which fed chutes leading to the bagging floor.

The jig concentrates were handled similarly. On starting up the plant, the first serious difficulty encountered was in the regrinding section, which could not handle the tonnage, when grinding to the degree of fineness required for good work in the flotation section. With the idea of lightening the load on the pans, the rolls were set hard up with considerable pressure

¹ "Flotation Process at Kiloc, N. S. W.," *Eng. and Min. Journal*, Dec. 7, 1912.

between them so as to act by choke crushing. This increase in the quantity of slimes produced was immaterial owing to the ability of the flotation plant to handle this product efficiently. This readjustment made some improvement, but not sufficient to make up the leeway, and finally coarse screens had to be put on the King screening machines, increasing the aperture from 0.015 in. to 0.024 in. This resulted in a product altogether too coarse for good work at the flotation section, a point which must be kept in mind when considering the results.



FIG. 34.—Grinding pans, thickener and top of Minerals Separation flotation plant at the Silver Peak mill.

The pulp as it left the regrinding section had a dilution of about 15 to 1 and before being treated at the flotation plant this was reduced to 4 to 1 in a Kyloe pulp thickener, the surplus water being returned by pump to the mill water-supply tank.

The treatment at the Minerals Separation flotation plant involved the use of heat, acid and two oils. *Eucalyptus amygdalina* was used as an emulsifying agent and a dark lubricating oil mixture to maintain the froth. The pulp was heated to 120°F. by live steam introduced into No. 1 agitating box. The consumption of steam was heavy, being equivalent to about

10 hp. (non-condensing) even when the initial temperature of the pulp was kept at about 90°F. by passing some of the mill water through the engine condenser.

The consumption of the different kinds of reagents varied considerably with the different kinds of ore, but the average was about as follows: Acid, 20 to 30 lb. per ton; *eucalyptus amygdalina*, $\frac{3}{4}$ to $1\frac{1}{2}$ lb. per ton; mineral-oil mixture, 2 to $2\frac{1}{2}$ lb. per ton. The sequence and method of using the reagents had to be altered from time to time to overcome difficulties that cropped up in the treatment.

Necessity for a Clean Water-Supply.—One of the early difficulties encountered was of a peculiar kind and is well worth recounting. The water-supply for the mill was drawn from a dam, fed partly by surface drainage and partly by water from the mine. For four years previous to the commencement of milling operations this water supply had not been drawn on and had simply evaporated and soaked away. A few days after starting the plant the water became discolored, first to a greenish yellow, then through yellow to a brick red. As soon as this reddish water arrived at the flotation plant the process went out of commission. The trouble seemed to be a total absence of aeration, which no amount of eucalyptus oil or other reagent could remedy. As much as 50 c.c. of eucalyptus oil was added to No. 1 agitating box at one time without producing any real froth or any signs at the back of the flotation spitz of the violent ebullition that is characteristic when the machine is in good working order. Occasional "blobs" of air came to the surface but nothing in the shape of a froth.

The red discoloration was due to a fine precipitate which on analysis seemed to be basic sulphate of iron. The precipitate was soluble in excess of acid, but the consumption made this remedy impracticable. The trouble was finally overcome, partly by keeping the mill solution in closed circuit and partly by manipulation of the reagents. Even with the closed circuit the loss by evaporation and leakage amounted to about 5,000 gal. per shift and this had to be made up with mine water.

This method of working led to another difficulty in the shape of scale and incrustations which formed chiefly in the Minerals Separation machine, but also in the screens and settler. In the flotation spitz the deposit took the form of sponge-like masses of crystals, while in the agitating boxes, in spite of the strong

scouring action of the impellers, it was like a bad boiler scale. Both types dropped off at intervals and blocked the valves and pipes in the machine. At one period the trouble became so acute that it looked as if the plant would be a complete failure. The deposits were first noticeable when the solution contained only 450 grains of salts per gallon of solution, and up to a certain point they increased with the density of the solution. The incrustation appeared to be fine sand and slime bound together with sulphate of lime and magnesia.

At the King screens the meshes of the wire cloth became badly blinded. After a week or 10 days the screening area became so reduced that the screens could not handle the tonnage, so the wire cloth had to be removed and cleaned by pickling in a weak acid solution and then scrubbing with stiff wire brushes. The changing of the cloth on this type of screen is quite a tedious operation, and much time was lost thereby.

Samples of scale were assayed and showed in the deposit from the flotation spitz 61 oz. Ag and 14.5% Pb and in the deposit from the agitating boxes 16 oz. Ag and 3.5% Pb. The residue after evaporating filtered solution assayed 4 oz. Ag and 1.2% Pb, showing that small quantities of both silver and lead were being dissolved.

Removing the Floating Froth.—The removal of the float from the surface of the flotation spitz presented another difficulty. When the machine was doing its best work the float was dense and heavy, being more of the nature of a thick scum than the usual frothy float composed of a fairly light mixture of bubbles and coagulated sulphides. The usual method in a case like this is to use paddles, but here they were ineffective as the float would not feed forward under the paddles. If the paddles were run fast and dipped deep, so as to cause a forward suction, the slime was stirred up and overflowed the lips with the concentrate, making it low-grade.

To overcome the difficulty the device shown in Fig. 35 was put in. It consists simply of a pair of ropes traveling forward along both sides of the spitz. A $\frac{3}{4}$ -in. pipe was run along the front of the spitz boxes, mounted on three simple bearings. To this pipe, 4-in. rope sheaves were keyed, two to each spitz. Along the back of the boxes a $\frac{3}{4}$ -in. iron rod was run and another set of 4-in. sheaves threaded on and held in place by pins through the rod so that they could revolve freely without

moving sideways. One-half-inch manila rope was spliced on over the sheaves. The $\frac{3}{4}$ -in. pipe was driven at a very slow speed by a long rope from a slow-moving countershaft in the mill. Different-sized driving sheaves were kept handy so that the speed of the rope rabbles could be altered when necessary.

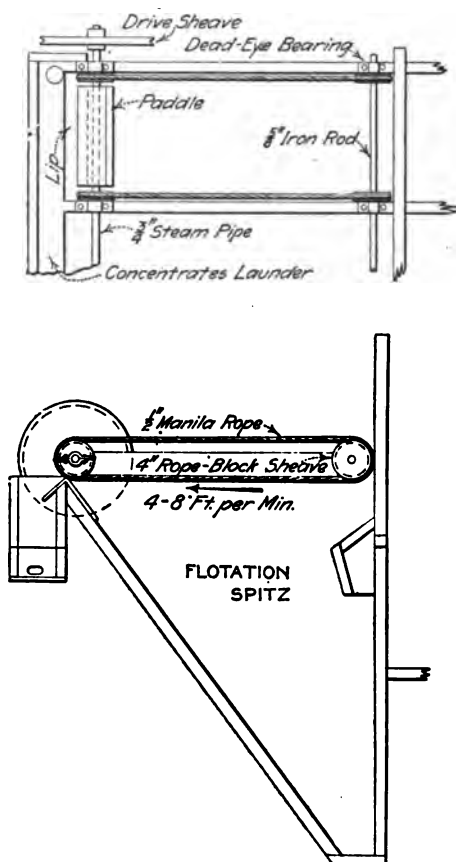


FIG. 35.—Device for removing foam at the Silver Peak mill.

It varied from 4 to 8 ft. per min. for the different classes of ore.

A coil of rope was kept handy, and a template was marked on a bench so that as the ropes wore they could be replaced rapidly by new ones. The ropes were put on with a rough long splice, and the men soon became expert enough to replace a rope

in a few minutes. While one rope was out of business, the other did the work fairly well. Once the amount of shrinkage had been determined and allowed for, this characteristic of manila rope caused no trouble, its hairy nature made it effective, and this, combined with its low cost, made it preferable to rubber or composition bands. After about 12 months, the back sheaves had to be renewed.

If the surface of the water was kept within $\frac{1}{4}$ in. of the lips these rope rabbles would feed the float into the launder, but as this close regulation of the water level was difficult, very small paddles were mounted between the sheaves on the front pipe so as to dip just behind the lips and touch the froth over.

The assays of the floats from the various spitz boxes are shown in Table 2. The overflow from No. 6 spitz was always, and that from 4 and 5 sometimes, returned to No. 1 agitating box for retreatment. A retreatment scheme for the whole of the concentrates in a separate machine, though highly desirable under the conditions, was not feasible—first, on account of the extra cost of plant and treatment, and second, because the amount of concentrates produced (from 3 to 5 tons per 24 hr.) was not sufficient to keep a machine adequately supplied.

Washing Concentrates to Improve their Grade.—In spite of all precautions the bulk concentrate from the machine ran about 15% insoluble. To improve this a scheme for washing the concentrates in the settling bins was introduced. The bin was fitted with a number of inclined removable baffles, and behind the first one a heavy spray of cold water was played on the float. This had the effect of disintegrating it so that the sulphides, still in the form of coagules, settled to the bottom at the same time a portion of the floated gangue was carried over a lip by the slow-moving current of water. The solids in the overflow assayed 50 oz. Ag, 10% Pb, and 70% insoluble.

The average value of the mill concentrates is shown in the table:

	Au, dwt.	Ag, oz.	Pb, %
In jig concentrates.....	6.0	69.4	21.5
In flotation concentrates.....	7.4	115.7	19.6

The mill tailings ran from 8 to 12 oz. Ag per ton, under 10 grains Au, and 2 to 2.5% Pb. The proportion of jig concentrates to flotation concentrate was about 2 to 1. The recoveries for the whole mill averaged only about 70% of the silver and

TABLE 1.—GRADING ANALYSIS OF REGRINDING SECTION

Grades	Energy unit	Feed		Discharge	
		Weight %	Total E.U.	Weight %	Total E.U.
- $\frac{1}{8}$ + 12	10.7	25.8	276.06		
-12 + 20	15.0	24.2	363.00		
-20 + 30	17.0	11.7	198.90	0.4	6.8
-30 + 40	18.0	6.6	118.80	4.0	72.0
-40 + 60	19.5	7.7	150.20	24.3	473.8
-60 + 80	21.0	4.0	84.00	15.5	325.5
-80 + 100	23.0	2.6	59.80	9.1	209.3
-100 + 150	24.0	1.4	33.60	4.9	117.6
-150	27.5	16.0	440.00	41.8	1,149.5
		100.0	1,724.40	100.0	2,354.5
					1,724.4
				Difference	630.1

Tons per 24 hr., 33.6.

Horsepower, 16.0.

$$\text{Efficiency} = \frac{630.1 \times 33.6}{16} = 13.2$$

TABLE 2.—ASSAY OF OVERFLOWS FROM FLOTATION BOXES

Box No.	Silver, oz. per ton	Lead, %	Insoluble, %
1	158.0	25.6	9.3
2	132.7	23.2	10.8
3	134.0	23.0	10.7
4	130.0	22.0	15.7
5	123.0	21.5	18.5
6	49.0	9.0	50.6

75% of the lead. In considering these low recoveries it must be remembered that during the period under review the plant was operating under two great disabilities—first, the inadequate regrinding facilities, and second, the fact that during the whole time, except for a few days toward the end, the plant was operated only one shift of eight hours per day. To give an idea of the importance of the former, a complete grading analysis of the feed and products from the flotation plant is given in Table 3. In columns 25 and 26 the recoveries made on each grade are given. Note the low recoveries on the coarser grades. It is not until the -100 to +150 grade is reached that anything like respectable recoveries are effected. The heads which remain on 80 screen contain 687.4 oz. Ag (by addition in column 5). The corresponding tailings contain 599.8 oz. (by addition

in column 21), which represents a recovery of only 12.7%. The heads which pass an 80-mesh screen contain 1,285.6 oz. Ag, while the corresponding tailings contain only 312 oz., which represents a recovery of 75.7%.

Had the whole of the heads been ground to pass an 80-mesh screen, the -80-mesh product would have been enriched by 687.4 oz., bringing the total up to 1,937 oz.; while, assuming that the extra tonnage of -80-mesh product would not affect the recovery made on that grade (if anything it should improve it), the -80-mesh tailings would be increased by 34.3% of 687.4 oz., or 235.8 oz., bringing the total loss in the tails up to 547.8 oz., which would give a total recovery of 72.2% of the silver.

The recovery under the existing conditions was only 53.8%, so that by grinding the whole of the feed to pass an 80-mesh screen the recovery would have been increased at least 18%, making the total recovery for the mill about 90% of the silver. Unfreed values were not altogether responsible for the losses due to the coarse crushing. Much of the free mineral itself was in pieces too large to be successfully floated. This was especially the case with the iron minerals and the finely crystalline galena. The large particles of coarsely crystalline galena appeared to float more readily. A good tailing of fairly coarse free mineral could be obtained by panning a sample of tailings which had been allowed to weather for some time so as to overcome the tendency of the sulphides to float off. So that better recoveries could have been expected if the sands had been passed over concentrating tables before being treated at the flotation plant. It would probably prove more economical to crush fairly fine, say not more than 15% on 80 mesh, and to table the sands, than to crush the whole product through 80 mesh and rely exclusively on flotation for the recovery of the sulphides. But here again, at this mill, the limited capital available was the governing factor.

The Daly West Mill.—The new mill of the Daly West Mining Co., Pratt City, Utah, has been described by L. O. Howard.¹

The mill, which went into commission in January, 1915, is housed in two structures, one 71 ft. long, 27 ft. wide and 60 ft. high from floor line to top of ore bins, housing the storage bins and the primary crushers. The mill is of the hillside type, the

¹ *Met. and Chem. Eng.*, Sept. 15, 1915.

TAILINGS
100 T 94.2 Tons (by Difference)

Grades by I.M.M. Screens	Weight of Heads %	Assay Values		Assay Values		Total Values per 100 Tons of Feed		Per Cent. of Total Values in Feed		Recoveries Made on Each Grade,	
		Ag, oz.	Pb, %	Ag, Oz.	Pb, %	Ag, Oz.	Pb, Tons	Ag	Pb	Ag, %	Pb, %
— 25 + 30..	0.4	12.0	1.5	12.0	1.4	4.44	0.005	0.22	0.17	7.5	13.3
— 30 + 40..	5.1	13.0	1.6	13.0	1.5	61.33	0.071	3.10	2.31	7.6	13.6
— 40 + 60..	23.4	11.5	1.6	10.8	1.2	290.16	0.332	15.16	10.93	8.1	25.9
— 60 + 80..	20.4	14.2	1.7	10.4	1.1	235.04	0.249	11.91	8.17	18.8	23.2
— 80 + 100..	9.4	17.3	1.8	6.5	1.2	66.30	0.122	3.36	4.02	59.2	27.8
— 100 + 150..	5.5	20.5	2.6	6.0	1.0	28.20	0.047	1.42	1.53	75.0	67.0
— 150	30.8	22.8	6.0	9.1	2.2	217.49	0.526	11.02	17.29	78.4	71.5
Totals.....	100.0					911.86	1.352	46.19	44.42		
Average totals		19.73	3.04	9.68	1.44					53.8	55.6

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fall from the head of the mill proper to the tailing launders in the tunnel beneath the mill being about 70 ft. There are five levels. On the top are the receiving bins, on the floor below are two Franz jigs, above which are the feeding trommel and Akins classifiers. On the third floor are the roughing Wilfleys, Harz jigs, Sherman tanks and regrinding rolls. On the second floor are eight concentrating tables and the flotation plant, and on the first, the Harz tailing jig and concentrate launders. The accompanying flow sheet shows the simplicity of the plant.

Milling the Ore.—The second-class ore is drawn onto an 18-in. conveyor which delivers to bins at the top of the ore house. From the bins the ore is sent in belt feeders over 3×6 -ft. grizzlies. A 10×20 -in. Blake crusher takes the oversize and discharges it into a trommel which is 48×72 in. and covered with double-crimped wire screen having two meshes to the inch. A 24-in. belt conveyor then takes the oversize of the trommel to a 48-in. Symons disk crusher. The head pulley of this conveyor is magnetized to remove all pieces of steel from the feed. The discharge of the Symons crusher and the undersize from the trommel are taken from the ore house to the mill bins by a 24-in. Robins belt conveyor, 210 ft. long with 6° slope. There is a Blake-Denison weigher having a capacity of 30 tons per hour.

The ore is sampled as it falls into the bin by a box rotating on a vertical axis at the rate of one revolution in 10 minutes. From the bin the ore is fed into a 48×72 -in. trommel covered with two-mesh screen and the oversize goes directly to one of two Franz jigs, the undersize going to the other after the slime has been removed in a 51-in. Akins classifier. The jigs are of the movable-screen type and are more often seen in the Missouri zinc district than in the West. There are screens 10 ft. long and 27 ft. wide. On the coarse jig the screen size is changed at intervals, the first two feet having $\frac{5}{16}$ -in. apertures, the next three feet having $\frac{1}{2}$ -in. apertures and the remainder being divided between screens having apertures of $\frac{5}{8}$ in. and $\frac{3}{4}$ in. respectively. The fine jig has $2\frac{1}{2}$ ft. each of $\frac{3}{16}$ -in., $\frac{1}{4}$ -in., $\frac{5}{16}$ -in., and $\frac{3}{8}$ -in. screens. The feed is divided into the proportion of 120 tons per day to the coarse and 80 tons to the fine. The coarse jig makes three finished hutch products averaging 30% lead which goes to the lead concentrate bin. The first two hutches from the fine jig are also finished products and will

average about 35% lead. The third hutch of the fine jig goes to two Wilfleys which make a finished zinc concentrate and a tailing that is retreated on the Harz jig on the lowest level of the mill. Tailing from the coarse jig is reground in 14 × 36-in. rolls and is elevated to a series of two sets of three impact screens where three sizes are made; on 4 mesh, through 4 mesh, on 8 mesh and through 8 mesh. The coarse is returned to the rolls. The middle size is treated on two three-compartment Harz jigs that make two finished hutch-concentrate products and a third hutch and tailing that are retreated on the same Wilfleys that receive the third-hutch product from the fine Franz jig. The undersize joins the slime from the Akins classifier and from the later table treatment, and is classified in a five-spigot Richards-Janney classifier. Each spigot feeds two Wilfley tables, the oversize after thickening going to the flotation plant.

The ten Wilfleys make similar products, a finished lead, concentrate, tailing and middling, which is retreated for a finished zinc concentrate and a tailing on three Wilfleys, and head water which is returned to the classifier. The concentrate from the Harz jigs and Wilfley tables drop to separate lead and zinc launders suspended on hangers running across the width of the mill in a tunnel on the lowest level. The head end of the launders is connected with a shaking device that causes the concentrate to flow to two sets of elevators in series, which discharge into two Akins dewaterers, the drained concentrate dropping into lead and zinc bins. They are then trammed in 1-ton cars through a tunnel to a sub-level of the ore house and are then elevated on a cage to the conveyor shed connecting shaft and ore house and trammed to the shaft. They are then lowered to the 1,200-ft. level and hauled out through the Ontario tunnel to Park City where they are loaded on wagons and hauled to the sampling works.

Flotation Applied to Slime.—The feed to the flotation plant consists of slime from the Richards-Janney classifier, which is thickened in one large and 23 small settling tanks of the type used in the old mill. The unit consists of two mixing cells and 14 flotation cells having a capacity of 30 tons of dry slime per day. The type is similar to that used by the Federal Lead Co. Each cell is 4 ft. 4 in. long by 20 in. wide and 5 ft. 8 in. deep at the tailing-discharge ends. They are set side by side with the sides in common. The shape of the cells is indicated diagram-

matically in the accompanying sketch. The slime is fed into the first mixing chamber where it is mechanically agitated by a vertical propeller driven by a belt from a line shaft extending the length of the unit. From this compartment it is drawn into a second similar one and then into a third, which is part of the first flotation cell. Each flotation cell has two compart-

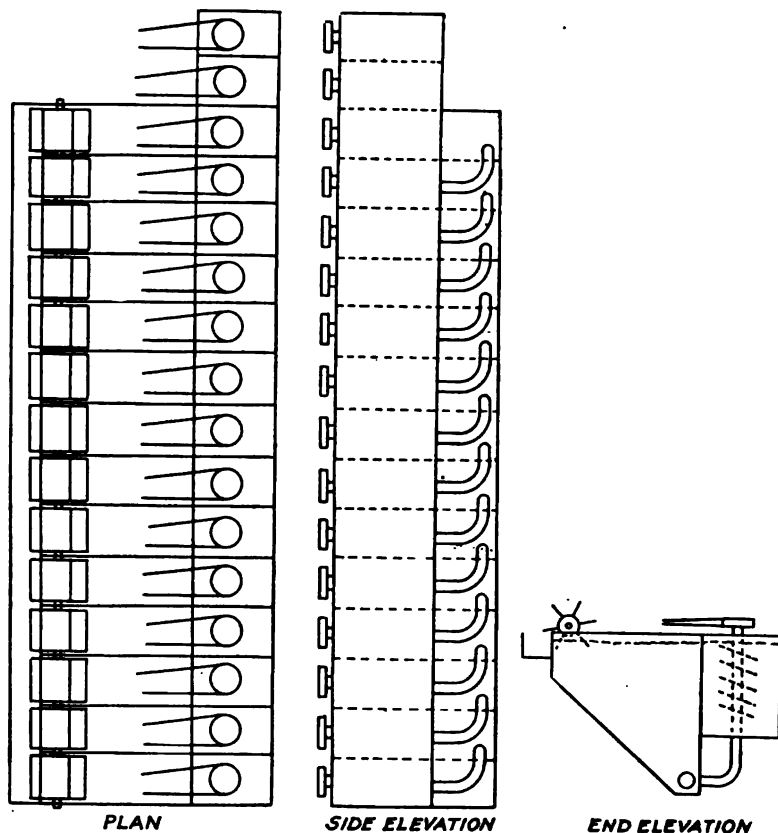


FIG. 37.—Diagram of flotation apparatus at the Daly West mill.

ments, one similar to the mixing cells and another larger one where the flotation is effected. A flotation concentrate is taken off each cell, the tailing passing through the whole system of cells in series.

The propeller in one cell draws the tailing from the bottom of the preceding flotation compartment and discharges through a

slot in the top into its own flotation compartment. The froth floats to the opposite side of the flotation compartment and is skimmed off by a paddle wheel that drops it into the concentrate launder where a jet of water breaks up the froth. The concentrate flows to a centrifugal pump that lifts it to an elevator delivering it to concentrate bins. This concentrate is marketed as a lead product.

The tailing settles to the bottom edge of the flotation compartment and is drawn into the next cell and again subjected to flotation, until it has passed the final cell when it goes to the tailing discharge. Flotation tailing is sampled by dipper. The weight of the material treated is ascertained by specific-gravity determinations made on a large sample caught during a definite period. Pine oil is introduced drop by drop into the first mixing cell. The consumption of oil is said to be about 50 gal. per month, which is equivalent to about 0.4 lb. per ton of slime treated.

The mill is operated one 8-hr. shift daily with a force of 15 men, and treats 200 tons per shift. Operations are controlled by means of the data obtained at the following points: The weight of ore fed to the mill is obtained from the Blake-Denison weigher, and the analysis of the feed is made on the sample taken at the end of the conveyor belt that brings the crushed ore to the mill from the crusher or ore house. Weight of concentrate is known and analyses are made on samples taken by pipe sampler from each car as it is taken from the concentrate bin. Analyses of tailing are made on the sample taken by tilting-box sampler from the discharge of the tailing jig. The tailing from the flotation department, which does not go to the tailing jig, is sampled by dipper and the weight obtained by specific-gravity determinations made on pulp discharged for a known period. Regular samples are not taken during the course of ore through the mill, but in case of any irregularity in the general tailing assay, the suspected machines are subjected to check sampling in order to remedy the trouble.

The compact flotation system has replaced the complicated classification and retreatment of slime and has done away with 12 slime tables. There still remain a large number of dewatering tanks preceding the flotation tanks which are used to thicken the feed to it.

CHAPTER XI

FLOTATION OPERATING PLANTS

The flotation process, either with or without oil, is now operating either experimentally or commercially in almost every mining camp in the United States and Canada. A large number of plants have adopted flotation, either as a principal process or as an auxiliary, and with many others it is only a question of a short time when the process will be utilized as a step in the scheme of metallurgy. It would be hopeless to attempt to explain the



FIG. 38.—Mill of National Copper Co., National, Idaho.

procedure at all operating plants, but it will be of interest to outline the methods followed at some of the important and successful works to which flotation has proved a valuable aid.

National Copper Co.—The National Copper Mining Co.'s mill, near Mullan, Ida., contains the first commercial, large-scale application of the Callow pneumatic flotation system. In this plant, stage grinding, table concentration, regrinding, and flotation concentration is practised. The accompanying flow sheet shows the details of the practice followed. Other plants using

A plant at the Iowa mine, in Colorado, has also installed the Callow type of flotation machine. The box is 2 ft. 4 in. by 9 ft. inside, the slope of the diaphragm compartment bottom is 4 in. to the foot. There are seven compartments into which hose lead from the air line. The sand discharge is regulated by a cock, but will be changed to a plug, the wear being excessive on metal parts. The water level is regulated by a short pipe turning on a loose elbow in a separate box on the discharge end of

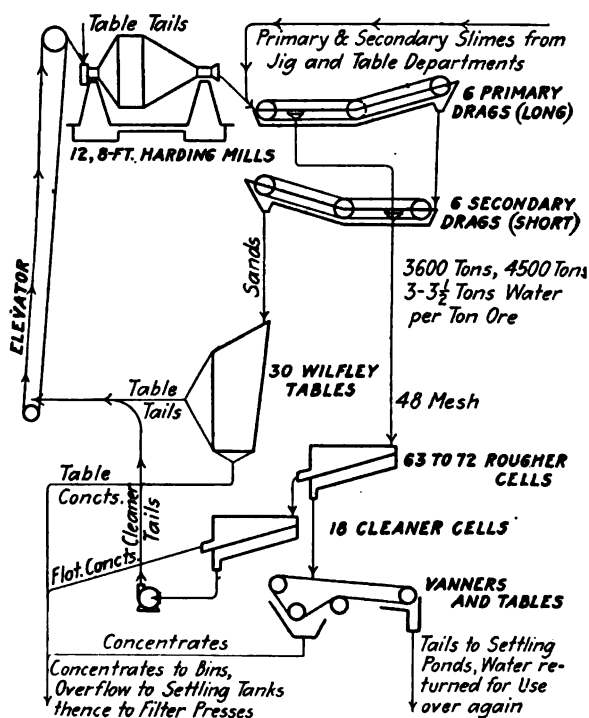


FIG. 41.—Flow sheet of Arizona Copper Co.'s plant at Morenci, Ariz.

the machine. The air-line pressure is about 6 lb. per square inch. The mill practice provides for primary crushing in a Blake crusher, then through rolls and jigs (three-fourths of the concentrate product is drawn off the screen on hutches). Jig tailings are dewatered by a screw conveyor and are all sent to the chilean mills and crushed through a 30-mesh screen. This product is classified and sent to concentration tables. The middlings are separated in spitzkasten and returned to one table for retreat-

ment. The slimes all go to a large settling box, where the top is skimmed and a flotation product is taken below the top. This passes to a pump where oil is introduced and then into the feed box on a machine.

Pine oil and creosote are used. Pine oil alone will work on some of these ores, while on others an excess of pine oil is detri-

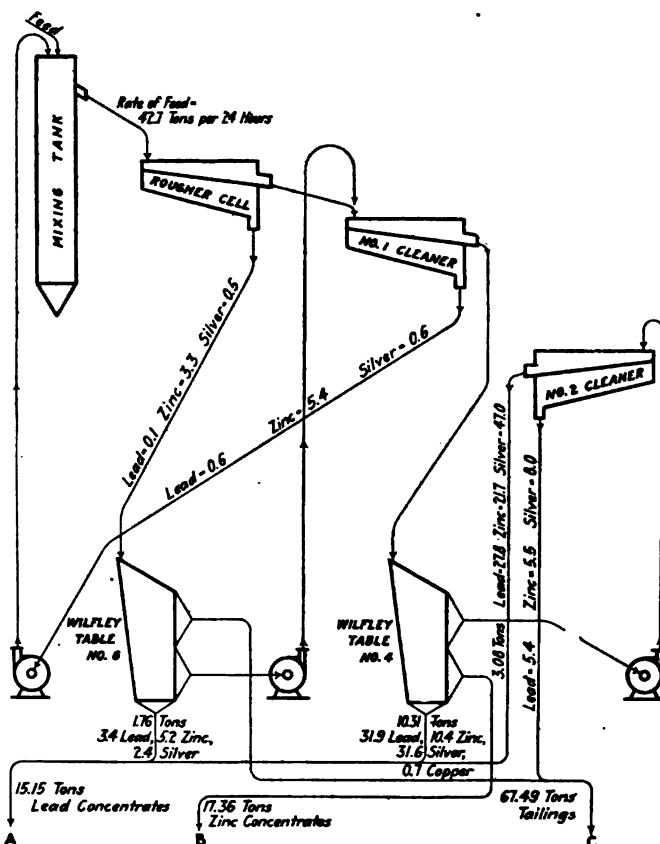


FIG. 42.—Flow sheet of installation at the Daly Judge mill.

mental, but creosote will work very satisfactory alone. The ore treated at the Iowa mill comes from Silver Lake basin and is the result of extraction by lessees on the Iowa and Silver Lake mines.

Gold King and Atlas Mills.—At the Gold King mill the whole plant is now on a flotation basis and is doing excellent work along the lines laid out before, which may be seen from the accompany-

ing flow sheet. At Ouray, the Atlas mill is crushing through a gyratory breaker first to $2\frac{1}{2}$ in., then through stamps, 1,500 lb. using 12-mesh screens, and then to Callow cones. The spigot product goes to Wilfley tables with special riffles, while the overflow goes to the flotation department. There are two Callow cones which thicken to 50% material which is then reground in an 8-ft. by 22-in. Hardinge mill, the discharge going to another two Callow cones, from which the spigot portion is taken to Wilfley tables again, and the overflow to flotation. Concentrates from the table go through a drier and are shipped. The flotation machines are of the Minerals Separation type, in which the froth from the first and second cells go into the concentrates. Froth from the last five compartments is skimmed and returned through the machines.

The discharge from the flotation goes to launder-type classifiers from which the overflow is taken to tailings and the spigot product to Deister sand tables that make dirty concentrates. They take everything up to 50% silica, and this is all put over the Wilfley table. This last step is used as an indicator to ascertain what the flotation plant is doing, the result showing up readily on the table deck. The oils used are fuel oil, pine oil, and pine-tar oil in the ratio of three parts, two parts and one part, respectively. No. 1 can be replaced by any light lubricating oil; the mill has successfully used winter black. No. 2 can be replaced by turpentine and No. 3 by creosote. An interesting feature has developed in the use of these oils. They formerly mixed up a supply for one day only. At one time an oversupply was mixed by accident and allowed to stand. On using this combined oil an increase in efficiency was immediately noted. The explanation which has been made is that fuel oil and pine oil, or turpentine, by some chemical action precipitates free carbon which breaks down the froth. For this reason its efficiency is increased by standing whereby the carbon is precipitated, and now they mix a week's supply at one time. Trouble is experienced with colloidal silica, and, for this reason, the tailings from flotation must be set over tables.

The Oneida Stag Mill.—A recent article¹ by R. W. Smith shows the practice followed by the plants where cyaniding was displaced by the flotation process. This is at the Oneida Stag Mines and Mill Co., at Idaho Springs, Colo. The ore is said to

¹ "Flotation Replaces Cyanide," *Eng. and Min. Journal*, Jan. 15, 1916.

be typical of the Idaho Springs district. It has been milled for the gold and silver contained in pyrite, gray copper, chalcopyrite, and a little galena. In the early days, the ore from the mine was high grade, and was then stamped, amalgamated, and concentrated. Various systems of metallurgy were tried at this plant for many years, but thoroughly satisfactory results were not obtained until after experiments had been made on flotation, which finally replaced the cyanide process formerly operating at the mill.

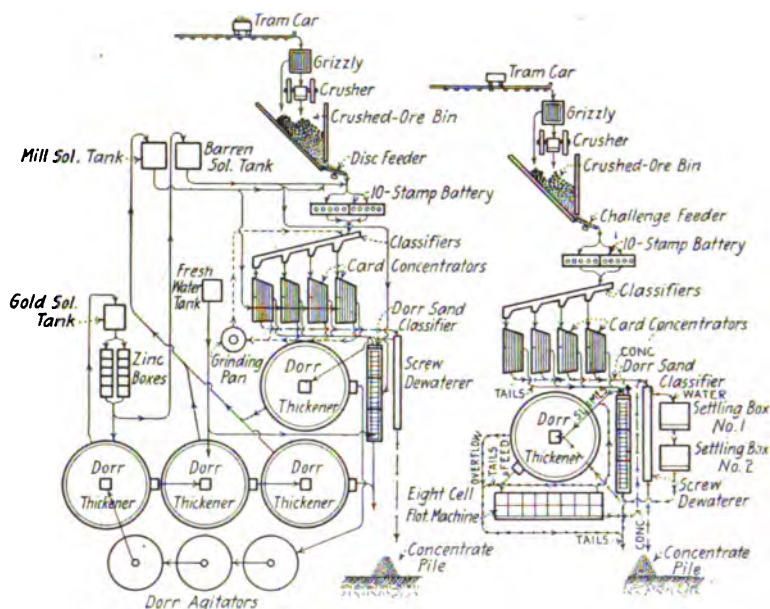


FIG. 43.—Flow sheets of the cyanide plant and the flotation plant into which it was changed.

After a thorough season of experiments the cyanide part of the mill was changed into a flotation plant. The accompanying drawing indicates the flow sheet of the original cyanide plant and of the flotation plant replacing it. The Dorr thickener and Dorr classifier shown were both part of the cyanide plant and fitted perfectly into the flotation scheme. The flotation machine itself was built in Denver and shipped in one piece to Idaho Springs. Two weeks after it arrived the feed was turned in, and in a little over three weeks it was making a commercial profit.

The entire tailing from the table floors is run through the Dorr

classifier, which eliminates the coarse sand. This is advantageous because the coarse sand contains practically no value, only about 14c. in gold and 15c. in silver, and by removing it, the wear of the flotation machine is considerably reduced. After removing the sand, the slime is conducted to the Dorr thickener, as shown, where it is dewatered. The thickened pulp, as discharged from this machine, is elevated to a small box near the first cell of the flotation machine by means of a diaphragm pump. From here the feed enters the flotation machine. The oil is added at this point, but neither acid nor heat is necessary in the treatment.

The flotation machine has eight cells, or compartments, connected in series, through which the pulp is made to flow by an external method of circulation of special design. The agitation is accomplished by a double agitator driven by a quarter-turn belt for each cell. The agitators in each cell turn in the same direction, since they are both fastened on the same shaft, but the blades are turned at such an angle that one impeller opposes the action of the other. The peripheral speed of the impeller is about 1,500 ft. per minute. The agitator cells are 18 in. across inside the lining. The tailing discharged from the eighth cell is reconducted to the Dorr thickener in which the slime from the sand classifier is thickened. This is an innovation that increases the recovery materially.

Handling the Concentrate Froth.—The froth from all eight cells is conducted to a screw dewaterer, into which is also conducted the table concentrate. This is advantageous because the table concentrate is coarse and stacks easily and is readily handled. If a special product is made of the flotation concentrate, an expensive dewatering system is required and it may involve an expensive filter press. With the mixing system, the froth goes directly with the table concentrate into the dewaterer. A total of 100 lb. of slime concentrate per hour is the amount delivered by a flotation machine. The dewaterer removes about two-thirds of the total tonnage of the slime, with the coarse concentrate. The outflow from the dewaterer carries away the other which is then led to two large settling boxes.

The overflow from the second settling box is led back to the Dorr thickener, thus forming a closed circuit and none of the concentrate is lost. Nearly all of the remaining third of the slime concentrate that overflows the dewaterer settles in the

settling boxes. The insoluble material raised with the mineral is considerable—about 16% in the first cell and about 30% in the eighth, or last cell, making an average of about 24%. There is only a slight penalty for this, and therefore there is not much objection to it. The table concentrate is much cleaner, and averages about 9% insoluble. After mixing in the flotation concentrate, the total product averages about 16% insoluble, an amount for which the penalty is quite low.

Table concentrate assays about $1\frac{1}{2}\%$ copper, and as the buyers do not pay for less than $1\frac{1}{2}\%$, the amount is lost. The flotation concentrate, however, runs about $3\frac{1}{2}$ to 4% in copper, and an increased profit was believed to be available by making a separate production of the flotation froth. After considering the matter, however, it was found that there is no difference, since $1\frac{1}{2}\%$ of copper is lost whether it is in the flotation concentrate or in the table concentrate. If the two are mixed, the assay is lower in copper, but the tonnage is greater, and this offsets the higher assay but lower tonnage of the flotation froth.

At the first suggestion to return the flotation tailings to the Dorr thickener, in which the flotation feed is thickened, it seems that there were several things that would defeat the purpose. In the first place it might be supposed that the pulp would build up until the thickener would not handle it properly. Or, if this did not happen, it might be that by crowding the thickener with more pulp, the additional pulp would crowd some of the valuable minerals over the overflow, and they would go to waste. The pulp does gradually build up in the thickener, but this is overcome by allowing the flotation tailings to run to waste through the tailrace one hour in every eight. The other effect is a slight increase in the value of the thickener overflow, but a marked difference in the value of the overflow product and the normal discharge. The flotation tailing averages about 40c. in gold, and the same in silver.

The overflow from the thickener averages about 40c. gold and about 20c. silver. Thus 20c. additional saving per ton is gained by this procedure. Considering that the total value of the flotation feed is about \$2, this means an additional saving of 10%. When the tailing from the flotation machine is added to the thickener, the feed to the flotation machine contains six parts of water to one of ore. When this tailing is discharged directly to waste, the consistency is 9 to 1. When the tailing is returned

to the thickener, the value of the overflow is about 60c. in total gold and silver. When it is bypassed to waste the average value of total gold and silver is about 55c. The difference is due to the silver content. Thus, for seven hours out of the eight the tailing from the mill is two products, the first of which is coarse sand from the classifier, having an average value in total gold and silver of 55c., and the second of which is the overflow from the Dorr thickener, with total value in gold and silver of 60c. The remainder of the time the tailing is composed of three products, of which the first two are the same as those just mentioned, and the third is a discharge from the flotation machine, having a total value in gold and silver of 80c. per ton. Using these figures as a basis on a \$2 feed, the flotation recovery is about 70% of its own feed. The additional saving made by this machine in connection with recovery made by tables, makes the mill total recovery about 87 to 90%. This is satisfactory considering the low content of the crude mill feed.

Of the flotation feed 85% will pass through a 200-mesh screen and about 90% of the flotation concentrate will pass through the same screen. The oil is added in a small box under which the diaphragm pump discharges just before the pulp enters the machine. Oil is added at the rate of $2\frac{1}{2}$ lbs. per ton of dry feed. The cost has been materially reduced by mixing three different kinds of oil.

In this case the oil mixture consists of one part of thick crude wood tar—fairly cheap—one part of spirits of turpentine—more expensive—and four parts of a cheap local mineral oil. This mineral oil constitutes a cheap base for carrying the other oils. It is not always possible to find a mixer that will give the high recovery and at the same time make clean products. By mixing oils as is done at the Oneida Stag, the cost per gallon for oil on the ground is reduced from 45 to 23c. This reduces the cost of oil per ton of crude mill feed from 6 to 3c. It has been estimated that the total cost of treatment, including flotation, is \$1.85. Of this flotation costs 35c. The accompanying table shows a list of average assays of products in various places as indicated.

In comparing the former cyanide practice with the present flotation plant work, the advantages of the latter are striking. An effective comparison is made by considering the two flow sheets. In the flotation mill the discard included three Dorr thickeners, three Dorr agitators, an air compressor for the agi-

VALUE OF VARIOUS FLOTATION PRODUCTS

Sample	Au, oz.	Ag, oz.	SiO ₂ , %	Cu, %
Flotation feed.....	0.07	2.00		
Flotation tailing.....	0.02	0.80		
Flotation froth (total 8 cells).....	0.60	15.00	24.00	3.5
Flotation, first cell.....	0.70	17.00	18.00	5.0
Flotation, eighth cell.....	0.50	13.00	30.00	2.0
Table concentrate.....	2.25	8.00	9.00	1.3
Dorr-thickener overflow.....	0.02	0.35		
Sand from the Dorr classifier.....	0.02	0.30		
Mixed concentrates from dewaterer..	1.70	11.50	16.00	2.5

tator, and a motor to drive it, two zinc boxes, a melting room, solution tanks, and the necessary pumps. The amount of machinery represents considerable outlay, and the saving is material.

The next comparison is in treatment cost. Cyanide costs 80c. per ton of mill feed, or \$1.50 per ton of ore actually treated by cyanide. Flotation costs 35c. per ton of crude mill feed, or 66c. per ton of ore actually treated. The difference between the two, 45c. per ton, represents the total profit per ton of mill feed with the flotation machine. The difference in the cost of installation is another advantage in favor of flotation. The actual cost of changing the mill over to flotation from cyanide was only about \$2,000. This includes the flotation machine and all labor required to install and connect it properly. Flotation has put this mill back into a state of productiveness and opened up for it a future better than might have been the case with any other process or combination of processes.

Floating Flour Gold.—Another flotation installation¹ was described by this same writer, and since it is of interest, some excerpts from the article will be given here. An interesting question arose in the treatment of the difficult ore, presenting a common but undesirable problem. In the early history of the mine the ore ran high in gold and silver. Later, mixed sulphides of zinc, lead, and iron became dominant. The first concentrating mill of modern design was erected about three years ago, and succeeded in saving, to a certain extent, the zinc, iron, and lead sulphides, but the gold and silver recovery was not satisfactory. This mill was treating ore of an average assay of 0.3 oz. gold, 2.5 silver, 5% lead, 8½% zinc, 3½% iron, and 0.5% copper. The gold was free and so fine that panning would not show a

¹ "Flotation of Flour Gold," *Eng. and Min. Journal*, July 1, 1916.

color. No hand specimens of gold were ever seen in the crusher feed, although occasionally a small piece was caught on the jig screens. Since the gold was fine, it was necessary to grind the ore to an extreme degree, to liberate for concentration. Cyaniding was impossible commercially.

After grinding to 8 mesh, as much sulphide as possible was removed on Card concentrators, and one Harz jig. Tailing was then reground to -30 mesh, amalgamated on plates and reconcentrated on concentrators. At various times the scheme was alternated by using carpets in place of plates, the free gold from the first riffle on the table being amalgamated in an arrastre. The simplified flow sheet indicates the source and disposal of slime. The high maintenance cost of slime concentrators, together with the great amount of space occupied and the low saving they made rendered this arrangement undesirable. The difficult problem, however, still remained. The zinc concentrate from the table was unfavorable, since it assayed 0.75 oz. gold, 5 oz. silver 10% lead, 33% zinc, 10% iron, 2.2% copper, and 11% insoluble. In order to raise the percentage of zinc the Huff electrostatic system was employed. A plant was erected that would treat all the zinc concentrates from the wet mill, and under these conditions the saving did not exceed 55% to 65% on any mineral. With such a loss it was natural to seek a process that would make a better saving. Flotation was suggested, and since a few preliminary tests gave promising indications a systematic campaign of testing was inaugurated.

Early tests were run on entire mill tailings, which showed that a small quantity of salable product could be made, but the tailing operated on was so lean that 90% recovery did not justify an expensive installation. The average feed for these tests was -30 mesh and assayed 0.12 oz. gold, 2.3 oz. silver, 2% lead, 0.3% copper, 1.5% iron, and 3.5 % zinc. From this feed rough concentrate was recovered, and assayed 0.5 oz. gold, 2.8 silver, 3.1% copper, 10.1% lead, 12% iron, and these tests weighed heavily against an installation operating under such conditions. Acid would have necessitated a more costly plant, more floor space, more difficult operation, and additional boilers for generating steam, besides which the cost of acid alone would have been almost prohibitive.

Under these circumstances, the scheme of treating the entire tailing was abandoned, and testing was confined to the slime

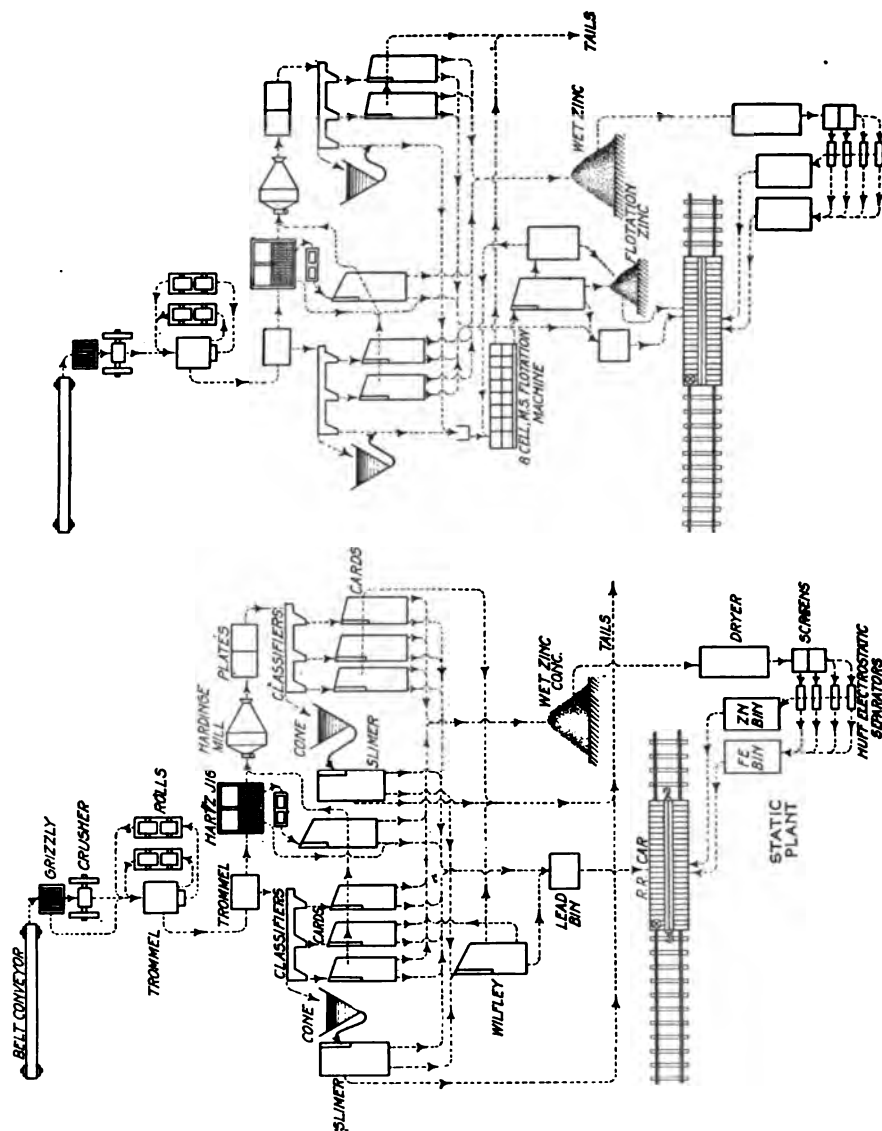


FIG. 44.—Flow sheets before and after the change to flotation.

alone. A small amount of clean lead, and a slightly greater amount of zinc was made from the slime by gravity treatment. The latter was dirty and had to be reclaimed on another table, riffled especially to treat the slime. The accompanying table shows some results obtained from this arrangement.

RESULTS FROM TREATMENT OF SLIME ALONE

	Au. oz.	Ag. oz.	Pb. %	Zn. %
Feed.....	0.14	2.3	4	6.3
Lead concentrates.....	3.00	8.8	38	15.0
Zinc concentrates (to be recleaned)...	0.70	6.0	12	18.0
Zinc concentrates (cleaned).....	0.35	7.0	8	36.0
Lead concentrates (from recleaning table).....	2.00	9.0	40	10.0

Slime concentrators received a total of 30 tons of feed each day, of which 90 to 95% was passed through a 200-mesh screen. The first series of flotation tests on these slimes were made with acid and heat. All normal variations of testing were tried and a saving of 80% was common on all minerals. Calculations showed, however, that the cost of acid was a serious item. Certain tests disclosed the possibility of simplifying and cheapening the process, and in this change a new set of conditions were introduced eliminating the acid, heat and any considerable pulp thickening. Products formed under these conditions assayed 0.65 oz. gold, 12 oz. silver, 18% lead, 35% zinc, low iron about 4% insoluble. On this product, however, further treatment had to be made to separate it into salable products. The high percentage of lead indicates that a fair separation might be expected. Furthermore, the froth was not hard to break up. A commercial unit was installed, and as soon as it was put into operation better results were obtained than in testing. Froth concentrate, as removed, was washed with clear water to a Wilfley table. The table classified the relatively coarse lead and zinc and made a perfect separation of them, but had very little effect on the remaining fine slime. The coarse lead and zinc products were heavy enough to settle were easily dried and handled. The difficulty was found with the fine products. The slime concentrate was pumped into a settling box, and the clear water overflow conducted back to the flotation machine. The products were dried only enough to allow them to be stacked. Steam at boiler pressure was used in a coil. The list of assays are given in the accompanying table.

RESULTS FROM FINAL TREATMENT SYSTEM

Sample	Tons per 24 hr.	Au. oz.	Ag. oz.	Pb. %	Zn. %	Fe. %
Flotation feed.....	30-25	0.14	2.5	3.0	5.5	1.0
Flotation tailings.....	28-30	0.03	0.8	0.8	1.5	
Flotation froth (total).....	4-5	0.65	10.0	15.8	36.0	
Flotation zinc (coarse), after table treatment.....	2-3	0.50	9.0	5.0	54.0	4.0
Flotation zinc (fine) after table treatment.....	1-2	0.50	12.0	11.0	40.0	5.8
Flotation lead, after table treat- ment.....	½-1	3.40	11.0	75.0	5.0	

One of the first things noticed in the operation of the flotation machine was the fact that the free gold floated. The greatest gold values were in the first riffles of the Wilfley table, although the lead content extended over many more than did the high gold value. It was also shown positively that the lead did not contain any marked amount of gold. At times there was a small accumulation of gold in spots on the table that had become greasy, and then the flour gold could readily be seen. The percentage of saving made reached about 85% gold of the gold.

The machine cost approximately \$1,000, including material, labor, and designing. The peculiar advantages of flotation here are that no acid is used, no artificial heat, no additional pulp thickening devices are necessary. Froth is easily broken down after removal from the machine, no classification is necessary before concentrating on the table, and the saving of all valuable slime content is raised from 20 to 75 or 80%. The process is simple in operation, produces better products than were ever before possible at the mill, and no slime losses occur in dust. The static plant was relieved of undesirable slime, and there was no royalty to pay as with the static process. One man per shift only was required to operate the plant, which will run successfully on a lean slime feed. Not only are the actual mill results extremely favorable, but there are other considerations that tend to make the whole property more valuable. Among these may be mentioned the impounded tailing dump of approximately 50,000 tons carrying gold, silver, zinc and lead that can be retreated at some future date. Lower grade ore can be sent down from the mine, as the recovery of concentrate per

ton of feed is higher with the flotation installation. Many tons of low-grade ore from the mine dump can now be profitably treated. And last, the possibility of radical changes in the milling system are open, which changes might result in lower treatment cost of better products and higher recovery. This might be accomplished by laying more stress on the flotation and giving less work to the static mill. Although this installation was more or less experimental, it actually paid for itself and the test in two weeks of operation. Inasmuch as the installation has demonstrated that flotation can be used commercially to recover gold, this plant has shed further light on modern milling practice that will be welcomed by the engineering fraternities.

FLOTATION OF ARIZONA COPPER ORES

FIG. 45.—Flow sheet of sulphide copper plant at Magma mill.

described by W. C. Browning, manager for the company, all occur in a porphyry-filled fissure vein. Three classes of milling

¹ From "Notes on Flotation, 1916," by J. M. Callow, a paper presented at the February, 1917, meeting of the American Institute of Mining Engineers.

ore are being mined, the first being a copper sulphide ore which contains varying amounts of bornite, chalcopyrite, pyrite and chalcocite, impregnated or ribboned through a gangue of altered, siliceous porphyry or altered diabase. The second is an oxidized ore from the upper levels of the mine, which contains malachite, chrysocolla, and at times a small amount of cuprite, native copper, chalcocite and covellite. This ore is an oxidized product of the sulphide ores. The third is a zinc sulphide ore, containing the black form of sphalerite, galena, pyrite and a small amount of chalcopyrite. This ore occurs in separate shoots from the copper ores. The gangue is usually a very siliceous, altered porphyry. Both copper and zinc ores usually carry an ounce of silver to the per cent. copper or zinc.

LABOR DISTRIBUTION AT THE MAGMA MILL

	Total men
Ore sorting and coarse crushing, 1 shift of 12 men.....	12
Table floor, 3 shifts of 2 men	6
Flotation, 3 shifts of 1 man.....	3
Filter pressed and concentrate settling tanks, 3 shifts of 2 men.....	6
Shift bosses, 3 shifts of 1 man.....	3
Ore loaders, 1 shift of 4 men.....	4
Watchmen, 2 shifts of 1 man.....	2
Total operating.....	36
For repairs, general cleaning up, tailings-dam upkeep:	
Carpenters.....	2
Head repairmen.....	1
Repairmen helpers.....	5
Roustabouts.....	16
Sampler.....	1
Total.....	61
which gives 3.6 tons handled per man.	

Distribution of Power:

	Hp.	Shifts
Sorting plant.....	45	1
No. 64½ Marcy mill.....	45	3
5 × 12-in. tube mill.....	45	3
Concentrating machinery and miscellaneous.....	80	3
Flotation (15 cells).....	80	3
Pumping water.....	15	3
	6,720¹	

which equals 22.8 kw.-hr. per ton.

"Motor" horsepower-hours—not net horsepower-hours.

The accompanying drawing, Fig. 45, shows the flow sheet, which comprises a combination of hand sorting, gravity and flotation. The $6 \times 4\frac{1}{2}$ -ft. Marcy mill, here used, was the first machine of this type installed commercially. Some early troubles were experienced from overspeed and too much pulp dilution. Since then it has given excellent service. The pulp density is now 60% solids and the revolutions, 22 per minute.

As first installed, flotation followed gravity, with Dorr tanks for thickening the feed in the interval. The principal defect of this arrangement was the large amount of pulp storage ahead of flotation, so that adjustment of the oil, when fed in the tube mill, could not be followed in the flotation plant. This was changed, and flotation was sandwiched in between the first and second tabling. The lower cells are still retained in operation as "gleaners." These gleaner cells, at the present time, contribute something less than 5% to the total recovery and take care of any irregularity in the operation of the main plant.

The distribution of labor is as shown in the accompanying table.

The ratio of water to ore in the tailrace is about $7\frac{1}{2}$ to 1.

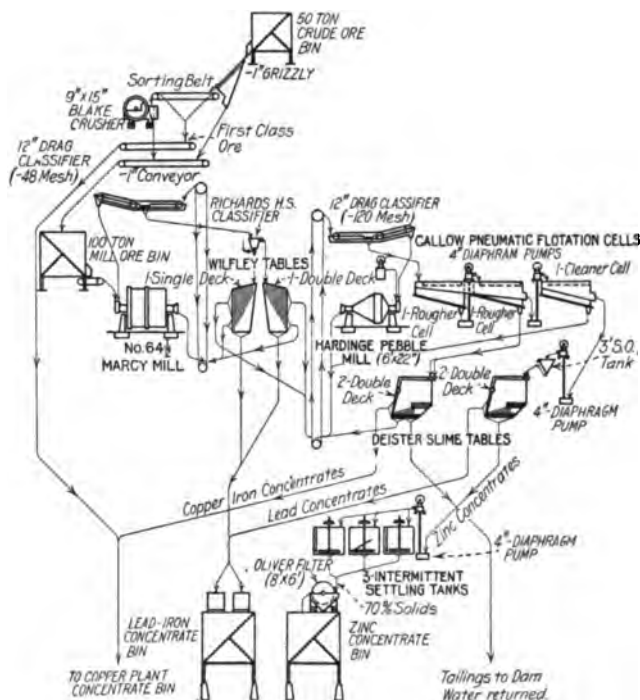
The mill was designed for 150 tons per day. On occasions it handles 250 tons per day. The original laboratory tests on this ore promised an 86% recovery on a 5.21 feed to the mill.

Part of the flotation concentrates are now settled by Dorr continuous thickeners and part in intermittent decantation tanks.

Magma Zinc-Plant Flow Sheet.—The flow sheet, given in Fig. 46, is substantially the same as the one at the sulphide copper plant. The main difference is that a drag classifier is used in closed circuit with a Marcy mill instead of a Callow screen. This mill crushes to 48 mesh instead of 10 mesh, as in the sulphide mill. A 6-ft. Hardinge mill, used for the final crush, is loaded with steel pebbles. The final crushing is now to 150 mesh. The plant was designed for 50 tons a day, which it will handle with ease.

For commercial reasons, the plant is now temporarily treating about 75 to 100 tons per day of sulphide copper ore, with results that are fully up to the standard for the sulphide copper plant. The only change necessary was to change the launders to the tables handling the zinc flotation concentrates. With the addition of the two more cells, it is expected to raise the tonnage to 150 tons per day.

This ore was first tested by the General Engineering Co. with a combination of roasting and magnetic separation, gravity and flotation. This gave excellent results, but the process was complicated and costly. Persistent experimenting with flotation eventually led to the present flow sheet.



[FIG. 46.—The flow sheet of the zinc plant at the Magma mill.

MAGMA COPPER CO., SULPHIDE MILL RESULTS
October, 1916

	Average tons per day	Assay in copper total oxides		Recoveries, per cent.	
Hand-sorted ore.	31.035	15.500		38.9	= 92.7%
Mill feed.....	187.565	4.042		53.8	
Mill concentrates	50.670	13.165		7.3	= 88% of mill feed
Mill tails.....	136.895	0.660	0.11	100.0	
Crude ore.....	218.600	5.672			= loss
September, 1916					
Hand-sorted ore.	17.830	18.000		28.90	= 93.39%
Mill feed.....	188.590	4.173		64.42	
Mill concentrates	49.563	14.400		6.61	= 90.02% of mill feed
Mill tails.....	139.027	0.529	0.14		
Crude ore.....	206.420	5.672			= loss

MAGMA COPPER CO., SULPHIDE MILL COSTS

September and October, 1916

	Costs per ton September	Crude ore October
Ore sorting.....	\$0.2130	\$0.230
Coarse crushing.....	0.0879	0.150
Marcy mill.....	0.0790	0.087
Tube mill.....	0.0637	0.086
Callow screen.....	0.0143	0.010
Tables.....	0.1840	0.200
Flotation.....	0.3210	0.345
Filter pressing.....	0.0746	0.072
Total.....	\$1.0375	\$1.180

These costs include all items of direct labor, supplies, maintenance and power, but no general expense or overhead charges.

It was found that No. 34 gravity fuel oil in combination with General Naval Stores No. 17 oil had the property of floating the zinc in preference to the iron. These tests have been substantially duplicated in practice.

Some trouble was experienced at first in making both the desired grade of zinc and tailings. Frequent changes in the grade of ore and the percentage of iron and zinc, owing to unavoidable conditions at the mine, made it difficult at first to get the desired results. It was also found that returned water from the sulphide mill pond contained sufficient coal-tar products to float the iron, thereby reducing the grade of zinc. Trouble was experienced at first with the oils which had lost some of their volatile constituents, due to hot weather. After correcting these conditions, results steadily improved.

The use of a small quantity ($\frac{1}{10}$ lb.) of copper sulphate was decidedly beneficial, reducing the tails from 2 to 3% zinc without materially affecting the grade of the concentrates. Prior to the use of copper sulphate, acid was tried and, while it lowered the tailings, it also lowered the concentrates. The dilution was 6 to 1 and was an important item in maintaining the grade of concentrates. The cells were sometimes run in parallel and sometimes in series. No differences in results were noticed.

The mill proper was operated with two men per shift. As it was found necessary to operate the sorting and crushing plants

only a few hours per day, no labor was employed here regularly; two men were taken from the crushing department of the sulphide plant for this work. Some iron and copper was hand-sorted at this plant and, together with the iron concentrates from the -150-mesh tailing tables, was shipped with the regular product of the sulphide plant. The intermittent settling system, as indicated in the flow sheet, was decidedly successful, delivering a 70% solid feed to the filter with a clear overflow from the tank. A summary of results is given in Table 4, showing the difference in results with and without copper sulphate.

The Sulphide Filming Plant.—The process in use in this plant is covered by Schwarz' U. S. patent No. 807501, which is the first disclosure of the use of a soluble sulphide for converting an oxide of a metal into a superficial sulphide, and afterward recovering it by a flotation process.

Earliest experiments were made with H_2S gas as the filming agent. A plant of 25 tons daily capacity was built. In this the gas was applied to the pulp by introducing it into the bottom of an open tank, having a mixing agitator. The results were encouraging, but the consumption of gas prohibitive—as much as 8 or 10 lb. per ton. The ore treated was the tailings of the Magma sulphide mill, which at that time carried considerable oxide. Occasional recoveries of 60% (of the total copper) were made, but the results were erratic owing to the difficulty of getting uniform filming of the pulp with this method of applying gas. Then followed an interval of several months when sodium sulphide, calcium sulphide and calcium sulpho-hydrate were used in an endeavor to avoid the use of gas, on the assumption that it was objectionable, owing to the danger of its poisoning the surrounding atmosphere. This was true when attempting to use it in an open tank, much of it being lost in the atmosphere. During this time a number of theories were advanced, exploited and abandoned. One of these was that natural and artificial sulphides could not be floated together, and that H_2S interfered with the flotation of the natural sulphides. The experience now is that H_2S , in proper quantity, promotes flotation of natural sulphides in company with the filmed oxides, and also that it is immaterial whether the oiling is done before or after filming, either plan is merely a matter of convenience.

In treating Magma sulphide tails, in which the principal losses were sulphides, the introduction of the gas not only filmed

MAGMA ZINC-PLANT RESULTS WITH AND WITHOUT COPPER SULPHATE

Without copper sulphate, Oct. 1 to 19, 1916	Zinc	Iron	Lead
Heads.....	15.38	7.80	3.89
Zinc concentrates.....	39.55	7.96	8.16
Lead concentrates.....	12.43	22.60	20.04
Tails.....	6.50	4.62	0.81
With copper sulphate, Oct. 20 to 31, 1916			
Heads.....	12.50	8.80	3.50
Zinc concentrates.....	38.50 ¹	9.16	6.57
Lead concentrates.....	15.93	24.01	12.60
Tails.....	3.23	4.56	0.78

¹ The lower zinc concentrates in the latter period were due to lower heads and not to use of copper sulphate.

the oxides present, a goodly percentage of which was recovered, but it also raised an entirely new crop of refractory sulphides.

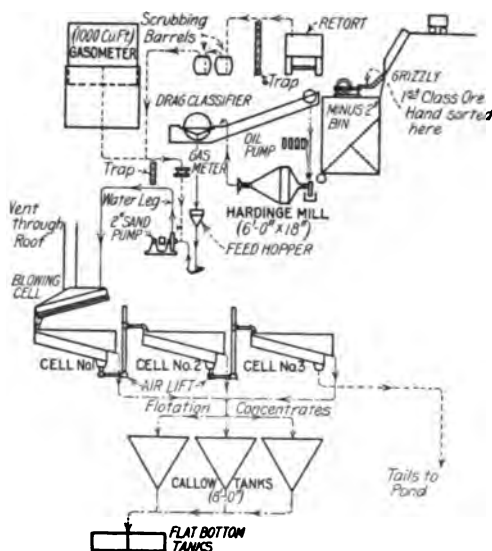


FIG. 47.—Flow sheet of sulphide filming plant at Magma mill.

An all-sulphide sample of regular Magma ore was tested with and without H_2S . The results are given in the table.

The commercial plant was shut down, and laboratory work was again reverted to, which resulted in the adherence to H_2S in preference to any other agent on this particular ore, and also to a complete change in the method of applying it. The plant was

reconstructed, crushing machinery and more cells added in accordance with the flow sheet shown in Fig. 47 for the purpose of treating Magma oxidized ores on a commercial scale.

The gas method used at present introduces the gas into the suction of a centrifugal pump in the manner indicated in the flow sheet. This has proved effective, greatly reducing the gas consumption, giving more uniform recoveries, and has removed all danger from the poisoning of the atmosphere; in fact, the commercial results now being obtained date from the first use of this expedient.

EFFECT OF H_2S ON NATURAL SULPHIDES

No H_2S gas	Magma Sulphide Ore			
	Heads	Tails	Concentrates	Recovery
Test 1.....	2.45	0.22	11.82	92.70
Test 2.....	2.45	0.25	9.62	92.20
Average.....	2.45	0.235	10.72	92.45
With H_2S gas to excess				
Test 1.....	2.45	0.28	9.40	91.30
Test 2.....	2.45	0.20	8.41	94.10
Average.....	2.45	0.24	8.90	92.70

The present method of making H_2S gas is to heat sulphur and oil in a retort. Various proportions of sulphur and oil have been tried, the present ratio being 1 of sulphur to $2\frac{1}{2}$ of oil. The temperature in the retort is kept at a uniform $300^\circ C$. At times the making of gas has given considerable trouble, owing almost entirely to changes from time to time in the quality of the oil used. Satisfactory results have been obtained at all times with California crude oil, but Texas oil always gives trouble, making a gas containing what is believed to be hydrogen-persulphide, which interferes with flotation and can always be identified by its eye-burning properties. This is in a measure overcome by careful scrubbing. Sulphur is precipitated in the scrubbers with free H_2S liberated.

Free gas in a pulp is fatal to flotation, hence the use of the blowing cell at the head of the first flotation cell. Experiments indicated that heating the pulp slightly before adding the gas is beneficial.

The cost of manufacturing the gas by this method will of course vary greatly according to local conditions. Those at Magma are abnormal. Sulphur is costing nearly 3c. today, and

oil nearly 1c. per lb., f.o.b. Superior, and costs on a basis of 30 tons per day and maximum of 3 lb. per ton stand as follows: 90 lb. of sulphur at 2.74c., \$2.51; 225 lb. of oil at 0.914c., \$2.16; total \$4.77 = 15.25c. per ton, or approximately 5c. per ton for each pound of sulphur per ton required by the ore.

On Magma sulphide tailings the gas consumption varies from $1\frac{1}{2}$ to $1\frac{1}{2}$ lb. sulphur per ton. On strictly carbonate ore assaying 3 or 4% copper, 3 lb. is an average figure, and on the latest test with a mixed carbonate and silicate ore assaying from 4 to 5% copper, 2 lb. of sulphur per ton. The fuel required for heating the retort is almost negligible. There are no cost figures since so far scrap lumber left over from construction has been burned for fuel. The labor item is unduly heavy, as the retort is situated about 600 ft. away and one man has to be held in reserve for the purpose. He could as easily make gas on one shift for 500 tons per day. With sulphur and oil at moderate prices on a 500-ton scale, using 2 lb. of sulphur per ton, an estimate of total gas cost is as follows: 1,000 lb. of sulphur at \$45 per ton (2.25c. per lb.), \$22.50; 2,500 lb. of oil at \$1.75 per barrel (0.436c. per lb.), \$11; 1 man at \$4; extra fuel for heating retort and sundry repairs, \$2.50; a total of \$40 per day. Thus the probable cost is 8c. per ton of ore, or 4c. per ton for each pound of sulphur required for the ore.

Other methods were tried—one using powdered coal in a separate retort instead of the oil mixture, and another in which the oil, instead of being mixed with the sulphur, was dripped into the sulphur retort with a force-feed lubricator. The usual iron-matte and sulphuric-acid method was also tried, but none of these had anything to recommend it over the one adopted. Iron matte was never seriously considered as a commercial possibility on account of the cost of acid.

The flow sheet given in Fig. 47 is self-explanatory. The 6-ft. by 18-in. Harding ball mill has a capacity of 35 tons per day when loaded with balls requiring 35 hp., or 45 tons per day when loaded up to 50-hp. requirements and grinding to 83%—150 mesh.

The power requirements are distributed as shown in the table.

This is the equivalent of 37.6 hp.-hr. per ton, or 28 kw.-hr. per ton.

The plant is operated by four men for the three shifts, the

extra man on day shift crushing the ore and at the same time hand-sorting out any first-class ore there is.

POWER DISTRIBUTION AT MAGMA MILL

1 6-ft. by 16-in. Hardinge ball mill.....	35 hp.
1 8 ft. by 14½ Root blower, No. 1(400 cu. ft. at 5 lb.)	} 20 hp.
1 7 by 14 Dodge crusher (1 shift only).....	
1 2-in. centrifugal gassing pump.....	
1 4-ft. diaphragm pump.....	
1 10-in. belt drag classifier.....	
1 Oil feeder.....	} 55 hp
1 Ore feeder.....	

Magma Results Summarized.—The table of summaries gives the results of gassing the sulphide mill tailings, the feed in all these experiments being the feed to the lower cells, or gleaner cells, as they are described on the flow sheet. The soluble copper in these tests varied from 0.3 to 0.45%, and the average for the entire period was 0.35%. Attention is directed to tests Nos. 63A and 67, giving results on this feed without gassing; further comparisons of more direct kind are given in the table on screen analysis.

The results on strictly carbonate ore are given, and on a mixed carbonate-silicate-sulphide ore, in the tables shown herewith. The latter tabulation gives a complete record and shows the contribution that hand-sorting makes to the total recovery. The carbonate results, for the most part, were on the rejections from the hand-sorting done at the mine, but in all these results the heads referred to are actual mill feeds, and the effect of hand-sorting is therefore not shown. On the mixed ore, gravity tables on the flotation tails would have added considerable to the total recovery, but on the straight carbonate ores they would have served no purpose. The question of the use of gravity-concentration machinery on flotation tailings here receives some light, but not enough to permit formulation of definite rules.

SCREEN ANALYSIS

Screen size	Heads				Tails			
	% Weights		Assays, % Cu		% Weights		Assays, % Cu	
	% indi.	Total Cu	Oxide Cu	Sulphide Cu	% indi.	Total Cu	Oxide Cu	Sulphide Cu
+ 60	8.3	0.63	0.16	0.47	10.8	0.490	0.14	0.350
+100	25.6	0.77	0.29	0.48	25.6	0.420	0.21	0.210
+200	20.8	0.75	0.35	0.40	24.4	0.350	0.16	0.190
-200	44.8	1.32	0.67	0.65	39.2	0.310	0.17	0.140
Total...	100.0	1.00	0.44	0.56	100.0	0.368	0.17	0.198
Assay...		1.00	0.44			0.350	0.17	0.180

EFFECT OF H₂S ON MIXED SULPHIDES AND OXIDES IN MAGMA SULPHIDE TAILINGS

No. H ₂ S	Heads	Tails	Concentrates	Recovery
	0.86	0.67	9.86	23.70
	0.78	0.60	9.63	24.70
Average.....	0.82	0.635	9.74	24.20
With H ₂ S	0.81	0.30	5.05	67.11
	0.78	0.29	5.51	66.38
Average.....	0.79	0.295	5.28	66.24

In all these tests pine-tar oil was used, the tonnage rate was 25 per day and the sulphur consumption $1\frac{1}{2}$ lb. sulphur per ton of feed; the soluble copper was 0.3%. The concentrates "with H₂S" are rougher concentrates.

The arrangement of plant did not permit of cleaning the filmed concentrates. The concentrates "no H₂S" were cleaned in the regular way.

SUMMARY OF RESULTS IN TREATING SULPHIDE MILL TAILINGS BY FILING, MAGMA COPPER CO., SUPERIOR, ARIZ., 1916

Test No.	Concentrates				Tonnage oils, % weight						Retort mixture		Remarks on gas, etc.	
	Heads	Tails	Rougher	Cleaner	Recovery ¹	Rate	C.T.	C.T.C.	F.O.	P.T.O.	C.P.O.	S		O
56	1.10	0.64	8.25	12.10	65.0		60	40	..	..	...	23	77	Excessive heat on old gas charge, test 57.
57	1.32	0.96	3.60	12.00	31.0		60	40	..	..	...	23	77	
62B	1.15	0.88	10.50	14.00	25.0		60	40	..	..	...	23	77	
C	1.15	0.69	10.50	14.00	43.0		60	40	..	..	...	23	77	Moderate heat, 1st firing, 0.77 lb. S per ton.
63A	1.40	1.21	11.90	18.30	15.1		60	40	..	..	...	23	77	No gas used.
B	1.40	0.96	12.90	15.10	34.0		60	40	..	..	...	23	77	Same retort as 62C, 2d firing, 0.36 lb. S per ton.
C	0.74	0.58	5.10	8.40	24.4		60	40	..	..	...	23	77	Same retort, 3d firing, poor gas.
64	1.24	0.69	5.40	9.80	50.9		60	40	..	..	...	23	77	Gas from iron matte.
66	0.91	0.63	7.00	8.20	33.2	29.40	60	40	..	..	...	23	77	Gas from iron matte.
67	0.58	0.47	5.00		21.0	29.40	60	40	..	..	...	23	77	No gas used.
68	0.83	0.49	6.70	12.50	44.2	31.20	60	40	..	..	...	23	77	Slow continuous firing, trying effect of oils.
69	1.16	0.83	12.20		30.9		..	40	60	..	...	23	77	Slow continuous firing, trying effect of oils.
70A	0.94	0.34	7.90	15.10	67.1	28.80	55	36	..	9	...	23	77	Slow continuous firing, trying effect of oils.
B	0.66	0.37	5.00	15.70	58.1	21.60	60	40	..	..	...	23	77	Slow continuous firing, trying effect of oils.
71B	0.91	0.44	7.90	15.70	54.5	31.00	55	36	..	9	...	23	77	Slow continuous firing, trying effect of oils.
72	1.08	0.47	7.20	11.50	60.5	16.50	..	..	..	..	100	23	77	Slow continuous firing, trying effect of oils.
73	0.88	0.41	7.40	17.10	53.3	27.20	55	36	..	9	...	23	77	Slow continuous firing, trying effect of oils.
74	0.86	0.49	7.70		45.8	17.30	60	40	..	..	...	23	77	To try out different oil mixtures. ¹
75A	0.76	0.38	3.80	8.80	55.6	31.20	32	20	..	25	25	23	77	To try out different oil mixtures.
75B	0.76	0.31	3.50	12.30	65.8	31.20	30	45	10	25	..	23	77	To try out different oil mixtures.
75C	0.76	0.38	7.40		52.5	31.20	30	20	25	25	..	23	77	To try out different oil mixtures.
76	0.80	0.47	4.10	17.20	46.6	31.20	30	45	..	25	..	23	77	Trouble with gas.
77	0.69	0.29	3.80	12.00	62.6	24.15	30	45	..	25	..	23	77	Repetition under better conditions.
78*	1.00	0.35	4.85	14.80	70.1	29.30	30	45	..	25	..	23	77	Repetition under better conditions.
79	1.23	0.42	7.40	11.25	69.9	30.90	30	45	..	25	..	23	77	Repetition under better conditions.
80	1.01	0.49	4.85	12.90	57.3	27.30	..	..	..	..	..	..	..	But with overheated gas.

¹ Based on rougher concentrates only.
very little effect on results.

C.T. = coal tar. C.T.C. = coal-tar creosote. F.O. = fuel oil. P.T.O. = pine-tar oil. C.P.O. = crude pine oil.

* For complete analysis of heads and tails see Table 8.

* Pulp was already oiled, and additional oils had

C.P.O. = crude pine oil.

FLOTATION OF ARIZONA COPPER ORES 229

SUMMARY OF RESULTS IN TREATMENT OF OXIDE ORE BY FILMING AT MAGMA COPPER CO.

Date	Test No.	Tonnage rate	Heads	Tails	Concentrates	Ratio	Recovery
July 8.....	150	35.00	2.78	0.86	9.50	4.50	75.94
July 9.....	151	30.00	3.15	1.06	9.72	4.18	74.31
July 12.....	154	26.00	2.92	1.17	7.02	3.34	72.00
July 13.....	155	30.00	3.24	1.22	11.66	5.16	69.72
July 14.....	156	33.00	2.92	1.35	10.26	5.67	61.94
July 15.....	157		2.92	1.08	8.21	3.89	72.27
July 16.....	158	28.00	2.75	0.83	8.30	4.14	77.25
July 18.....	159	30.00	2.99	0.70	8.32	3.32	83.78
July 20.....	161	30.00	3.08	1.08	9.18	4.06	73.62
July 23.....	164	31.00	3.92	1.62	10.80	3.96	69.76
Average.....	...	30.33	3.06	1.09	9.34	4.22	73.08
Aug. 9.....	175	25.0	3.04	0.71	14.85	5.41	80.49
Aug. 10.....	176	30.0	3.63	0.81	16.07	5.40	81.82
Aug. 11.....	177	27.0	5.46	1.25	17.36	3.82	83.22
Aug. 12.....	178	25.0	5.46	1.80	22.74	5.72	72.81
Aug. 13.....	179	27.0	5.12	1.43	16.80	4.17	78.68
Aug. 14.....	180	25.0	5.04	1.85	16.91	4.72	71.09
Aug. 17.....	181	25.0	4.23	1.62	18.48	6.46	67.64
Aug. 18.....	182	25.0	11.66	2.65	26.50	2.65	85.76
Aug. 19.....	183	25.0	6.73	2.67	22.26	4.82	68.62
Aug. 20.....	184	25.0	8.35	2.45	20.67	3.09	80.12
Average.....	...	25.9	5.87	1.72	19.26	4.63	77.025
Sept. 25.....	185	28	4.90	1.33	14.8	3.77	80.0
Sept. 27.....	186	27	5.50	0.79	17.8	3.61	90.0
Sept. 28.....	187	30	5.00	1.50	18.1	4.74	76.4
Sept. 29.....	188	25	3.10	0.61	13.9	5.30	84.7
Sept. 30.....	189	24	2.80	0.54	13.2	5.60	84.1
Oct. 1.....	190	23	2.30	0.58	15.1	8.40	77.8
Oct. 2.....	191	24	2.40	0.68	14.4	8.00	75.0
Oct. 3.....	192	27	2.60	0.41	14.3	6.30	86.0
Oct. 3.....	192	27	2.70	0.36	14.9	6.20	89.2
Oct. 4.....	193		3.60	0.61	10.3	3.60	89.6
Average.....	...	26	3.49	0.74	14.68	5.55	83.28

Best results in July tests were obtained with a consumption of about 4 lb. sulphur and 10 lb. fuel oil per ton feed. Sulphur consumption on tests 181 to 184 inclusive was 6 lb. per ton. In the other August tests a minimum of 2.25 and maximum of 3 lb. per ton. The September and October results were obtained with a sulphur consumption of 2.8 minimum and 3.5 maximum per ton.

The Burro Mountain Concentrator.—The Burro Mountain Concentrator of the Phelps, Dodge Corporation, at Tyrone, N. M., uses flotation as a part of its ore-treatment process. The mill and its operation were described in an article by F. C. Blickensderfer, in the *Engineering and Mining Journal*, July 14, 1917. The following description of the flotation operation is abstracted from that article.

The ore is of two distinct classes. The east orebody (also called No. 1 division) is composed of a quartz porphyry and quartz monzonite porphyry, with an average grade of copper of 1.90%, of which 0.20% is copper in the form of oxide and carbonate, and the remainder chalcocite. The Chemung (also called No. 3 division) is composed of granite, felsite and small amounts of quartz porphyry, with an average grade of 1.90% copper, of which 0.09% is in the form of oxide and carbonate, the remainder chalcocite. Both ores contain small amounts of water-soluble copper compounds, requiring the addition of lime for protection of iron parts.

The Dorr classifier sands comprise about one-half of the Marathon mill product, the drag slimes about one-half, or a separation into equal parts. The sand is delivered to a stationary distributor, from which six Butchart tables receive and treat the uniformly divided feed. These tables are of the same type as the roughing tables, except that their riffing is suited to the treatment of fine sands.

The proportion of solids of the table feed is 35 to 40%. Three products are formed—final sand tailings, middlings and finished concentrates, the middlings being elevated and returned to the Dorr classifiers, the concentrates entering the main concentrates launderer, while the tailings, containing about 20% solids, are sent to Allen cones for dewatering.

The sand product contains 75% solids and 2% -200-mesh material. The cone overflow contains about 4% solids, most of which passes a 200 mesh. The cone is at once a good deslimmer and dewaterer.

The overflow of the cones is pumped by a centrifugal pump to the lower Dorr thickeners and subsequently treated on Senn vanners. The cone sand is sampled automatically and then discharged to the tailings launder.

The overflow from the Dorr drag classifier goes directly to the flotation machines of the Rork type. Each Rork machine is

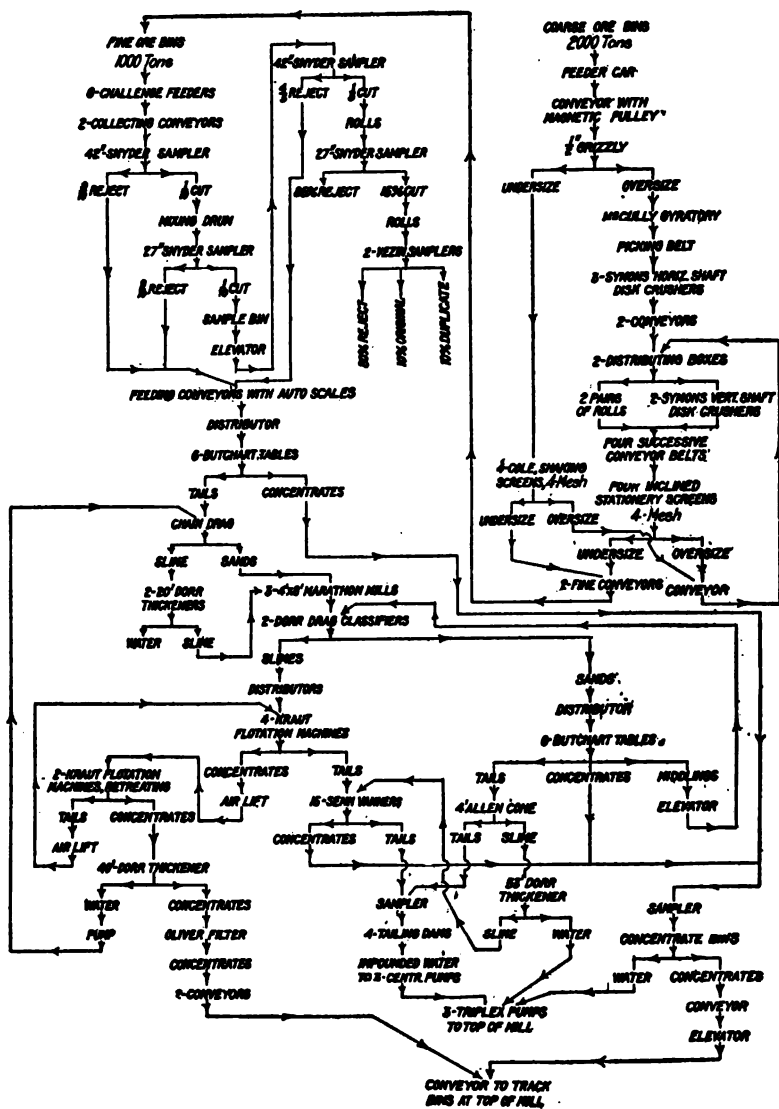


FIG. 48.—Flow sheet of Burro Mountain concentrator.

driven by a 15-hp. motor, connected by a flexible chain drive at a reduction of 109 to 17.

This type of flotation machine has the advantages of mechanical agitation, but at the same time it does not escape the drawback of a large amount of repairs and high power required.

Most of the oil is sprayed by steam jets into the Marathon mills, only a very small amount of light oil being added to the machines themselves. The Kraut & Kohlberg disk oil feeder is used to supply the oil. The oil storage is located in two 14,000-gal. horizontal tanks, beneath the roughing-table floor.

The flotation process has not been a pronounced success. It has been definitely determined that insufficient agitation is the primary cause of such difficulties. Recently the tailings from one Rork machine have been retreated in another Rork machine, and very good results have been obtained. This has been applied only to section A ore (Division 3 of Burro Mountain mines). The B section ore (Chemung) is of a different character and requires about 8 lb. per ton of commercial H_2SO_4 . Without acid the results obtained are poor.

The data shown in the table have been established by actual experiment.

FLOTATION DATA AT BURRO MOUNTAIN MILL

Tons dry feed treated by each primary cell.....	75
Percentage of solids in feed.....	20
Pounds oil per ton of feed treated by flotation.....	2.75
With an additional 8 lb. of acid for section B.	
Horsepower per ton of feed treated.....	0.20
Size of motor.....	15 hp.
Life of drum, about six weeks; if there are considerable amounts of sand in the feed, the drum will be cut out in five weeks or less.	
Flotation process will not work in a neutral solution.	
Section A ore is indifferent to acid, but section B ore demands it.	

The results, summarized since operation, are shown.

RESULTS OF BURRO MOUNTAIN MILL OPERATION

Copper in feed.....	1.90%
Oxide copper in feed.....	0.15
Ration of concentration.....	10:1
Extraction total.....	72%
Sulphides.....	77%
Fresh water per ton.....	800 gal.
Power consumed per ton.....	9.0 kw.-hr.

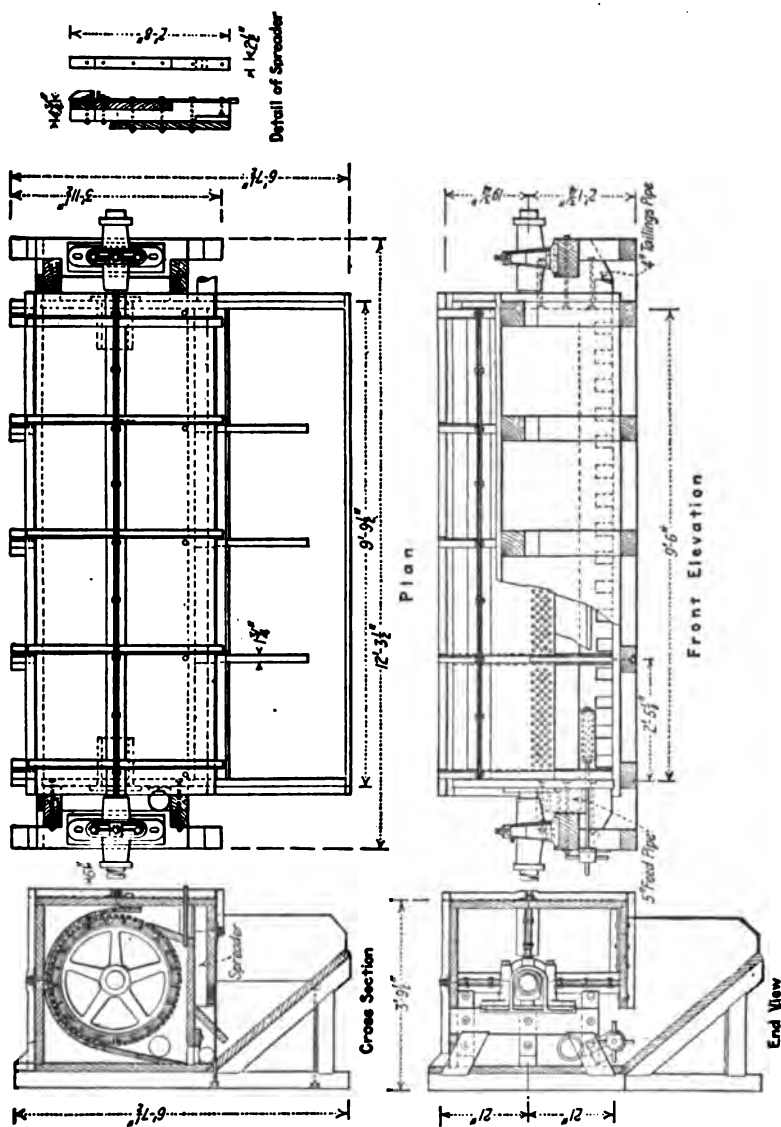


Fig. 49.—The Rork-type flotation machine.

In view of the fact that the flow sheet has been constructed with entirely new types of machinery, this concentrator may be considered a new departure.

Flotation at the Consolidated Arizona.—Although the Consolidated Arizona Smelting Co. of Humboldt, Yavapai County, Ariz., has conducted concentration of ores on a comparatively small scale, there are some features of this work that will be of general interest. The description is from an article by George M. Colvocoresses, in the *Engineering and Mining Journal*, July 14, 1917.

The concentrating ore comes from the two mines operated by the company—the Blue Bell and De Soto mines—and consists essentially of an intimate mixture of chalcopyrite and iron pyrites in a gangue of quartz and sericite, chlorite and hornblende schists (Yavapai schist). The average composition of the concentrating ore from the two mines is approximately as follows:

AVERAGE COMPOSITION OF CONCENTRATING ORES

	Blue Bell	De Soto
Au.....	0.035 oz.	0.03 oz.
Ag.....	1.40 oz.	1.30 oz.
Cu.....	3.20%	3.20%
Fe.....	15.00%	9.00%
S.....	17.00%	10.00%
CaO.....	1.00%	2.00%
MgO.....	1.00%	1.50%
Al ₂ O ₃	4.00%	4.50%
SiO ₂	57.00%	69.00%

The concentrator building was erected in 1905, intended at that time to treat only ore from the De Soto mine, which (as will be noted) is more siliceous, but the operations in 1905 and 1906 were not at all successful and only about 50% of the copper contents were recovered, the mill being closed down after treating a few thousand tons of material. The concentrator was started again in 1913, at which time the Blue Bell mine only was producing and 68% of the copper contents were recovered in the concentrator. Flotation was installed in 1914 and the De Soto mine was reopened in 1915. The main results of the concentrator operations during the last four years are given in the table following:

RESULTS OF CONCENTRATOR OPERATIONS

Year	1913	1914	1915	1916
Tons of ore treated.....	30,635 ¹	53,906	81,544	80,304
Average Au contents of ore treated, oz. per ton.....	0.08	0.0355	0.034	0.036
Average Ag contents of ore treated, oz. per ton.....	1.66	1.233	1.05	1.07
Average Cu contents of ore treated, %.....	3.70	3.504	2.83	3.19
Tons of Con. produced.....	10,462	12,281	19,877	24,514
Recovery of Au in conc., %.....	51.28	55.55	62.50	80.30
Recovery of Ag in conc., %.....	49.94	59.15	70.50	79.90
Recovery of Cu in conc., %.....	67.59	78.12	88.60	92.50
Ratio of concentration.....	2.93:1	4.29:1	4.1:1	3.28:1

¹ Includes 1,603 tons of Fennell ore with high gold values.

During 1915 and 1916 a certain amount of tailings from the old operations of the concentrator, averaging 1.75% copper, were treated and this reduced the average grade of mill heads and also tended to reduce the recovery of metals since the copper in the tailings is more or less oxidized and does not float as readily as the fresh ore from the mines.

During the latter part of 1916 and the beginning of 1917, new equipment was placed in the concentrator, and at present approximately 400 tons of ore is being treated daily with the expectation of treating between 500 and 600 tons in the near future as adjustments are made to the crushing equipment. The flow sheet of the concentrator has been altered and modified several times during the last four years and at present is represented by the accompanying diagram.

Some of the points that may be of special interest in connection with concentrating operations are treated under the following headings.

Fine Grinding Necessary.—Although for certain ores it is undoubtedly advantageous to take out as much mineral as possible in the form of a coarse product, an attempt to carry out this program has not been satisfactory at Humboldt. The minerals in the ore are intimately mixed with the gangue, and experience has proved that there is no thorough separation of the minerals and gangue until very fine grinding has taken place and until the particles of ore have been reduced to pass 65 mesh, or better,

tage. After the jigs were removed, all the feed was crushed to pass 10 mesh and a table product was made, and at present the table feed will practically all pass 20 mesh.

As the concentrator is now arranged, the discharge from the ball mills is classified and the overflow goes direct to the flotation machine, while the coarse material is sent to the two 8-ft. Hardinge pebble mills. The discharge from the pebble mills goes over the roughing tables, and the tails from the tables are returned to the classifier in closed circuit. The roughing table concentrates contain about 4.50% copper, 15% insoluble and 38% iron. This product constitutes about 60% by weight of the mill concentrates, but contains only about 30% of the copper recovered, so that even at this stage we still recover a far larger proportion of iron than copper.

The average flotation concentrates contain about 15% copper, 20% of insoluble and 25% iron, and the mixture of the flotation concentrates and the table concentrates gives general mill concentrates averaging in the neighborhood of 9% copper, 16% insoluble and 30% iron.

The typical grading and assay of feed of the flotation machines and tails therefrom are shown in the accompanying table.

GRADING AND ASSAY OF FLOTATION FEED AND TAILINGS

Screen size	Feed			Tailings		
	Wt. % solids	Assay % Cu	Wt. % Cu	Assay % solids	Assay % Cu	Wt. % Cu
On 65 mesh.....	7.5	1.18	3.4	7.2	0.56	12.7
100 mesh.....	11.5	1.62	7.1	12.5	0.38	13.7
200 mesh.....	20.4	2.33	18.0	22.2	0.22	13.5
Through 200 mesh.....	60.6	3.08	71.5	58.1	0.39	60.1
Total.....	100.0	2.05	100.0	100.0	0.39	100.0

Since the subsequent handling, roasting and smelting of concentrates are expensive, it is the aim to make as high copper concentrates as possible, and for this purpose the selective action of the flotation machine is valuable as showing a preference for copper over iron. On the other hand, the gold and silver values in ores generally follow the iron rather than the copper, and hence it is not desirable to lose the iron if in so doing an unduly high loss in precious metals is sustained, and, moreover, up to the

present time it has been found that a better recovery is made by the flotation plant if the heavy iron sulphides are removed from the ore before it is treated in the flotation machine.

These factors necessitate careful calculation and the accumulation of a considerable amount of data on which to base such calculations, and at present the operators are not prepared to state definitely whether it will pay to continue, as at present, to make roughing-table concentrates and then to follow this with the flotation machine, or whether ultimately it will be found advantageous to float the minerals first and afterward table the tailings for such coarse mineral particles as may be contained, as is at present the practice at Inspiration. The calculations are complicated by the metallurgical conditions at the smelter, where, up to the present time, these have always had an excess of silica, to offset which it has been necessary to purchase basic flux (either limestone or basic copper ore), and in the custom material purchased, 4c. cents per unit is paid for the iron and lime contained and the shipper is charged for all silica contained. It is therefore obviously an advantage to retain the iron in concentrates, provided that by so doing too low-grade concentrates, on which the smelting costs would offset the gain in fluxing value, are not made.

Crushing and Grinding the Ore.—Crushing and grinding has presented several difficulties, owing to the nature of the ore, which is exceptionally hard and tough. In starting out to enlarge the capacity of the concentrator, a jaw-crusher was used to reduce the ore to about a 3-in. ring and this was followed with a Symons disk crusher which was intended to deliver a product passing a 1½-in. ring, and the product to be delivered to a 6-ft. cylindrical ball mill and a 6-ft. Hardinge ball mill which were intended to reduce practically all the feed to pass 8 mesh, the same to be further reduced by Hardinge pebble mills in closed circuit.

The ball mills of both types proved entirely unequal to the task allotted to them, not through any fault of their own, but simply because impossibilities were demanded, and even the impact of a 5-in. steel ball failed to crack the larger pieces of ore, many of which went through the mills and came out as round unbroken pebbles the size of marbles. To remedy this fault, a 1-in. trommel was installed after the Symons disk crusher and the oversize returned, thus running the Symons machine in closed

circuit. This practice considerably increased the tonnage that the ball mills would handle, but resulted in unduly heavy wear on the Symons disk crusher, and eventually it was decided to install rolls to handle the oversize from the trommel and at the same time reduce the screen of the trommel to $\frac{3}{4}$ in. as used at the present time. The feed to the ball mills will now all pass $\frac{3}{4}$ in., and on this basis the ball mills are doing very good work and are crushing twice as much feed per day as could be passed through them when the 1-in. screen was in use, and it is the intention to install heavier rolls later and to reduce the feed to the ball mills to pass $\frac{1}{2}$ in., which will make a better arrangement.

The extreme hardness of the ore may be shown by another illustration, but in this second case it has been possible to utilize this hardness to good advantage. The fine grinding of the ore in 8-ft. Hardinge pebble mills was previously, for a short time, done with flint pebbles and the cost of the pebbles for grinding the very hard ore amounted to upward of 15c. per ton and would have been considerably more at the present higher price of pebbles. Gradually the ore from the Blue Bell mine was substituted for the pebbles, using blocks of ore about 4 or 5 in. in diameter, and choosing this material from the stopes in which the ore was hardest and heaviest. For a long time past no pebbles at all have been used, but the lump ore is fed into the Hardinge mills through the discharge end and the average consumption is slightly less than 9 tons of this ore per day, or about one-fiftieth of the feed to the mill.

The ore blocks as fed into the mills cost about \$2 per ton more than the crushed ore as delivered in the regular manner, but the mineral contained is, of course, added to the concentrates as these grind up and amply repay this extra charge, and therefore the net saving in using block ore instead of flint pebbles may be considered as 15c. per ton of ore concentrated.

Generally speaking, the commercial minerals in the ore float in an entirely satisfactory manner and when clean sulphide ore is sent to the mills, an excellent recovery is obtained, sometimes as high as 95 or 96% of the values, including the recovery made by the roughing tables. Various mixtures of oil have been tried out during 1916; the average mixture used had the composition shown in the table.

COMPOSITION OF FLOTATION OILS

67.7% Standard Oil Co.'s Stove Oil 27 plus
25.7% Pensacola Tar and Turp. Co.'s No. 200
6.6% other oils

100.0%

and on the average 1.387 lb. of oil per ton of ore was employed, the cost representing just about 3c. per ton of ore. At the present time the mixture of oil shown in the accompanying table is being used:

FLOTATION OILS NOW USED

70.4% Standard Oil Co.'s Stove Oil 27 plus
24.9% Pensacola Tar and Turp. Co.'s No. 200
4.7% Standard Oil Co.'s Calol fuel oil 24 plus

100.0%

and the amount is about 1.31 lb. per ton of ore and the cost about 2.5c. per ton.

Operating on the hard ore mentioned with a comparatively small tonnage and a mill that was in the first case poorly designed and that has been remodeled piecemeal, the working costs are naturally rather high; the best yearly record was made in 1915, when the total cost of milling was \$1.06 per ton exclusive of flotation royalty. The costs in 1916 were increased by the higher prices of material and particularly by the higher cost of labor, and the cost in that year was \$1.36 per ton, excluding flotation royalty. A division of the working costs for 1916 is, excluding flotation royalty: Crushing, \$0.391; grinding, \$0.471; tabling, \$0.262; flotation, \$0.240.

Improvements in Metallurgical Treatment.—During recent years the improvements in concentration practice, and more particularly the advent of flotation, have worked a great change in the smelting conditions and practice. Flotation has not only made possible the profitable treatment of many ores that could not be successfully handled by gravity, but as an adjunct to gravity concentration has greatly improved the recovery of metal and the grade of concentrates produced, and to meet this situation the nature of smelting equipment has been radically altered, the blast furnace being more and more replaced by

roasters and reverberatories. There are, of course, certain classes of ore that consist essentially of massive iron and copper sulphides, which naturally are treated in the blast furnace and for which no other means of treatment would be at all satisfactory; and conversely, such ores as are found in the porphyry coppers and in other mines with comparatively low-grade copper and high silica content can be treated only by concentration and subsequent smelting of the concentrates.

At Humboldt, however, much of the ore falls in what might be termed the "twilight zone" between concentration and direct smelting, and the decision as to whether certain portions of the ore produced should be concentrated or directly smelted in the blast furnace is a matter that can be decided only after careful calculation and the collection of a considerable amount of both milling and smelting data.

The smeltery at Humboldt is equipped with both blast-furnace and roasting and reverberatory plants, and very careful records as to the recovery and costs in both these departments must be utilized in deciding the best manner of treatment for portions of the orebodies that are neither very basic nor very siliceous. The very siliceous ore, coming largely from the De Soto mine and containing upward of 65% insoluble is, of course, a straight concentrating problem, and likewise the most basic ore at the Blue Bell, higher in gold and silver and containing in the neighborhood of 4% copper, 35% insoluble and over 20 units of iron and lime, constitutes an excellent blast-furnace charge. Between these two extremes the bulk of the production of the mine varies, and varying conditions, costs and recoveries in gold, silver and copper at the mill and smeltery determine just where the line between milling and smelting ore is to be drawn.

At present it is found better to concentrate all ore containing less than \$2 in gold and silver and less than $3\frac{1}{2}\%$ copper, where the insoluble exceeds 40%, which means that the iron will not exceed 16 or 17%. On the other hand, with the ore that is higher in gold, silver and copper than the foregoing and which (as usually follows) contains less insoluble and more iron, direct smelting is more profitable than concentrating.

As the development of concentration proceeds and lower working costs are attained with better recoveries also, the balance should swing in favor of concentration, but on the other hand, as conditions and equipment at the smeltery are improved at the

same time, factors in favor of direct smelting will be introduced, and the only way in which this question can be intelligently decided is by continually refiguring the calculations with the varying factors obtained from the latest cost and metallurgical records.

CHAPTER XIII

FLOTATION OF MISCELLANEOUS ORES

The Darwin Lead and Silver Mines and Development Corporation have in operation an experimental mill for the treatment of lead carbonates and sulphide ores from its several properties situated in the vicinity of Darwin, Inyo County, California.

The company's holdings consist of properties, the ores of which are widely different in character, both as regards the occurrence of the minerals and the gangues. Some of these consist of lead carbonate, carrying silver, disseminated through a hard jasper gangue, while others are a mixture of lead carbonate, galena and silver chloride in a gangue of hematite, quartz and calcite. They are mined from the upper levels down to a depth of about 600 ft., while the ores bellow the 600-ft. level consist of galena and some native silver in a calcite gangue.

The treatment of a lead silver ore by the Murex process, forms the subject of an article by H. S. Rexworthy in the *Engineering and Mining Journal*, July 14, 1917. The description is given here as a good example of the possibilities of the process.

A 25-ton Murex oil-concentration plant has been in operation for about six months, during which time ores from the various properties have been sent down for treatment. As already stated, these ores vary considerably, each one necessitating a separate study with a view to defining the best method of treatment and, contrary to general experience with this process, it has been found that the same reagents were in most cases applicable, and good results were obtained on the different ores, including dumps, carrying considerable amounts of fines of many years' standing.

The plant is now making recoveries of from 80% to 85% of the mineral content, treating lead carbonate carrying silver and a mixture of carbonate, galena and silver chloride in gangues of hematite, quartz and calcite. The success of the process has justified adding another unit, which has now been completed, making a plant capacity of 50 tons per day.

The mill is of the gravity type and the flow is as follows: Ore is drawn from the lower tramway terminal discharge shoot,

weighed and delivered upon a grizzly provided with 1-in. spaces. The oversize is then fed to an 8 × 12-in. Blake-Hendy crusher, breaking down to 1-in. ring and discharging into a bin of 100 tons' capacity. The ore is fed from this bin by means of a Challenge feeder into a 20-mesh, rotary, conical screen which passes the oversize, with an addition of water in the proportion of two to one of ore, into a 4 × 3-ft. Hendy ball mill, the discharge end of which is fitted with a screen delivering a 20-mesh product mixed with the undersize from the Challenge-feeder screen. The oversize in the latter case is returned to the ball mill for regrinding. By adopting this system a considerable amount of fines is eliminated from the ball mill. The pulp is then fed with a regulated mixture of oil and ground magnetite to two horizontal Murex tube agitators, each a plain steel cylinder 3 ft. diameter by 10 ft. long, revolving slowly, containing several hundred pounds of 1/4-in. iron or steel punchings. These become oiled with the mixture and collect the mineral particles both fine and coarse in the form of a paste which adheres to the punchings. This paste is continuously broken away by attrition and flows, with the pulp, through a slotted screen at the opposite end of the agitator, upon a brass shaking conveyor tray, which feeds it under a powerful electromagnet. The coated mineral particles are attracted by the magnet from the thin flowing stream of pulp to the undersize of an endless belt traveling at right angles to the shaking tray, thereby suspending them until out of the magnetic field, when they are washed off by water into a concentrates bin of large capacity, provided with a divisional bulkhead and end doors in which the concentrates are delivered and discharged alternately. They are finally hauled by tram wagons and sacked ready for shipment.

Deister Concentrator as a Pilot.—The overflow water from the concentrates bin and tailings together is passed over a Deister concentrating table. This is employed as a means of indicating to the millman any inefficiency brought about by bad coating or change in crude-ore feed, etc., at a glance. This can be seen by a small mineral streak appearing on the table. In such cases the operator makes the necessary adjustments in the oil-mixture feed to meet the new conditions.

The tailings finally pass to a large wooden spitzkasten where the coarse sands are settled out and the overflow discharges into four 20-ft. square settling ponds, from which the clear water

is returned to the mill, to be used over again. This water contains a certain amount of emulsified oil, and is found to be beneficial to the Murex process, since the oil can be made use of in further ore treatment.

An 80-hp. gas engine provides power for the mill and a 10-kw. d.c. generator current for the magnets and lighting of the camp. Also a 20-kw., 2,200-volt a.c. generator supplies power at the well, situated about 2 miles from the mill. The well pumps are electrically driven, of the vertical triplex type, provided with three 3-in. rams and 8-in. stroke and a working pressure of 450 lb. per square inch. Special electrical equipment enables this installation to be controlled from the mill.

In addition to the plant as described, a complete Murex laboratory outfit is installed in the mill, to be used when the various ores are sent for experimental treatment by the Murex process.

The Murex system differs from all other processes as it is not necessary to produce those conditions in which the oiled mineral particles will float. Practically all other oil-concentration methods can be equally well made by the Murex process. In addition, certain separations can be made that previously have been considered impossible. Thus lead, zinc and copper carbonates containing sulphides can be concentrated from their various complex gangues. Blende and galena can be separated from heavy spar and from pyrite. In the cases of some ores it has been found that material of 2- or 3-mm. section can be treated successfully by this system, provided the mineral is freed from the gangues, and the latter can be eliminated in one operation, leaving the mineral values in such a form as to be easily smelted.

Various blends of oils, such as crude petroleum and residuum, are used in proportions of $\frac{3}{4}\%$ to $1\frac{1}{4}\%$ of oil per ton of ore to which is added equal quantities or more of ground magnetite. This mixture of oil and magnetite forms a permanent paint which, no matter how finely it may be divided or broken up, will always be found to be composed of the two substances in the same ratio.

Considerable skill is exercised in blending the various oil mixtures so as to make them amenable to each individual ore to be treated. The thickness of pulp, quantity of water and method of agitation are all important, particularly in cases of differential treatment, as in separating lead from zinc.

MILLING AND MUREX-TREATMENT COSTS AT THE DARWIN PLANT OVER A PERIOD OF 30 DAYS

15 lb. oil @ 1.43c.....	\$0.2145
17 lb. magnetite @ 20c.....	0.2040
0.8 lb. oleic acid @ 15c.....	0.1120
Labor three men @ \$4.00.....	0.7500
Power and water supply.....	0.5000

Cost per ton..... \$1.7805

In the Murex process the ore need be ground only fine enough to free the mineral from the gangue. Both sands and slimes are treated together in one operation so that no classification is necessary, thereby reducing the crushing and running costs to a minimum.

Floating Antimony Ores.—The flotation of antimony ores has received comparatively little notice in the technical press, either because little work has been done along this line or because results have not been available for general information. Hoover¹ refers to antimony in the following statement: "Such minerals as antimony or molybdenum sulphides, which because of their friable nature suffer prohibitive losses in water concentration, are found amenable to treatment by the Elmore process."

An earlier reference in the *Engineering and Mining Journal*² is as follows: "We are informed that tests of the Elmore oil-concentration process on gold-bearing antimony ores have recently been made with very successful results. It is well known that antimony sulphide ores are very difficult material for concentration by ordinary methods, and that the treatment of such minerals carrying gold in Nova Scotia, Australia, Portugal and Austria has been wasteful and unsatisfactory."

Henry E. Wood, in 1912,³ refers to tests made by him and gives the first statement of results on an antimony ore:

TESTS ON ANTIMONY ORE (STIBNITE)

	Sb %	Recovery %
Crude, 100% by weight at 40 mesh.....	3.57	
Flotation concentrates, 6.50% by weight.....	47.40	86.30

¹ Hoover, "Concentrating Ores by Flotation," 1912 edition.

² Vol. 70, Nov. 3, 1900, p. 515.

³ Henry E. Wood, "The Wood Flotation Process," *Bull. A. I. M. E.*, November, 1912, pp. 1227-44; *Trans. A. I. M. E.*, Vol. 44, pp. 684-701.

Following the publication of these results, Donald G. Campbell¹ published the results of a study of oil flotation on an antimonial silver ore.

The investigation discussed in this article was undertaken at the College of Mines, University of Washington, by the authors² to determine whether low-grade Alaskan stibnite ore could be successfully concentrated to a product high enough in antimony to warrant the installation of a flotation plant. Paralleling this problem, a similar investigation was made on a gold-bearing antimony ore from the Nome district, Alaska. Partial results of the latter investigation were recently published.³ The significance of these tests is discussed in a later part of this article.

The ore tested in this investigation is a stibnite from the Fairbanks district, Alaska. The mineral occurs in lodes in a country rock of mica schist or quartz schist,⁴ which has been extensively fractured. The stibnite is found along the shear zones in kidneys and shoots and is associated with some accessory quartz. The mineral is finely granular and shows some oxidation phases that are red in color. This may be kermesite ($\text{SB}_2\text{S}_2\text{O}$).

The high price of antimony during 1914 and 1915 has led to the development of some of the Alaskan deposits. Brooks⁵ states that the shipments in 1915 from six mines were 833 short tons, valued at about \$74,000. The ore averaged 58% antimony. Of the six mines, four were in the Fairbanks district and two in the Nome district. The operations in the Fairbanks district were, however, on a small scale. Most of the ore was broken and concentrated by hand-sorting, and no ore carrying less than 50% antimony was shipped.

The sample of ore submitted for tests represented waste dump material, too low in grade to ship. It consisted of stibnite with some red antimony mineral, probably kermesite, in a schist gangue, which also carried quartz.

¹ D. G. Campbell, "A Study of Oil Flotation," *School of Mines Quarterly*, Vol. XXV, No. 1, November, 1913, pp. 28-34.

² Joseph Daniels and C. R. Corey, in *Engineering and Mining Journal*, Jan. 27, 1917.

³ "Flotation Tests on an Antimony-Gold Ore," Earl R. Pilgrim, *Engineering and Mining Journal*, Vol. 102, No. 19, Nov. 4, 1916, p. 820.

⁴ "Antimony Deposits of Alaska," A. H. Brooks, *U. S. Geol. Sur. Bull.* 649, 1916.

⁵ A. H. Brooks, "The Alaska Mining Industry in 1915," *U. S. Geol. Sur. Bull.* 642-A, 1916, pp. 29-30.

Analysis of the original sample is shown in the accompanying table:

ANALYSIS OF THE ANTIMONY ORE

Antimony.....	35.0%
Insoluble.....	41.2%
Sulphur.....	15.0% (estimated)
Arsenic.....	0.1%
Gold.....	0.14 oz.
Silver.....	2.00 oz.

The problem given to us to solve was the enrichment of the antimony content without regard to the gold or silver values.

All the tests were run in a Case laboratory machine which was operated at various speeds from 1,200 to 2,000 r.p.m. At 1,400 r.p.m. the peripheral speed was 1,190 ft.; at 1,500 r.p.m. the peripheral speed was 1,275 ft. Twelve hundred feet per minute was decided to be the best speed, and the machine was run in the final tests at 1,400 to 1,500 r.p.m.

Grinding and Pulp Densities.—In the first tests the material used was ground to pass a 65-mesh Tyler standard screen with an opening of 0.0082 in. The tailings carried visible mineral grains, and the next samples were ground to pass 80 mesh, opening 0.0069 in. These tests did not show a great improvement. To observe the effect of still finer grinding, an original sample of 80-mesh material was screened and a sample of the oversize on the 100-, 150-, and 200-mesh screens taken for test (experiments Nos. 28, 29, 30). The +200-mesh material gave excellent results. This was confirmed by a subsequent test (No. 33) in which a special sample was ground in a pebble mill to pass 150 mesh. For practical purposes, however, it was thought best to grind the material to pass 100 mesh (opening 0.0058 in.); all the later tests were made on this size.

Pulp densities of 1:4, 1:5, and 1:6 were tried. The 1:5 ratio of ore to water was selected, and the final tests were made on a mixture of 500 grams of ore and 2500 c.c. of water.

Temperatures varying from 10°C. (50°F.) to 30°C. (86°F.) were employed. Warming the mixture by steam to an average temperature of 26°C. (78.8°F.) was found to give the most satisfactory results.

The time of agitation varied from 10 to 30 minutes; 15 minutes proved to be sufficient to bring up most of the froth, and any excess time gave very little increase of concentrate.

The use of sulphuric acid was found to be necessary with the oils employed. Concentrated H_2SO_4 (specific gravity, 1.84) in quantity equal to 0.44% of the weight of the ore was finally adopted as the standard.

Various oils were tried, including pine, pine-tar, stove, acid sludge, pyroligneous-acid residue from Douglas fir distillation, gashouse creosote, wood creosote, and coal-tar creosote used for wood preservation. Wood and coal-tar creosotes were each found to give most satisfactory results. No special study of the character of the various froths was made, but the Canadian and the Japanese creosotes; either alone or mixed, gave firm froths and the best selection of sulphides. The amount of creosote oil for average tests was 0.60% of the weight of the ore.

The antimony content of the flotation concentrates was raised in all of the tests on the original ore. The lowest percentage of antimony was 39.6 (experiment No. 41), and the highest was 66.7 (experiment No. 1). Using creosote oils, the variation in content was from 50 to 65%. The percentage recovered for all tests varied from 54 to 92, but the average of the tests using creosote oils was from 60 to 90%. The corresponding tailings carried 8 to 20% antimony. Some tests in which the creosote oils were used with other oils showed a high recovery, this was due entirely to the creosote oil.

In order to determine whether retreatment of the tailings would increase the recovery, experiments Nos. 31, 32 and 34 were made on combination samples of tailings from earlier tests. These tests showed that 70% or more of the antimony could be recovered by the use of a secondary machine or by further treatment in the same machine.

A tabulation of the experimental work is given in the table.

Conclusions Drawn from Experiments.—The conclusions to be drawn from the work are as follows:

The low-grade stibnite ore can be successfully concentrated by flotation methods to yield shipping concentrates.

Fine grinding is necessary for the best recovery and highest grade of concentrate.

The creosote oils, with sulphuric acid, give the highest recoveries at the lowest cost of treatment.

Retreatment of primary flotation tailings will increase the total recovery.

A summary of the results of the investigation described in

the article, "Flotation Tests on an Antimony-Gold Ore," by Earl R. Pilgrim, is of particular interest in connection with the tests described in this article. For purposes of comparison a tabulation is made in the second table.

Although these tests were made at the same time and were constantly compared and discussed while the work was going on, they were run independently of each other. How far the personal equation entered into the conduct of the experiments is difficult to say, but the results, after the assays and calculations had been made, could not have been influenced by this factor. Were one to draw generalizations, it might be said that Alaska stibnite ores are amenable to successful concentration, using a small amount of sulphuric acid and a varying amount of creosote oil, at temperatures of 15° to 20°C. The ore should be ground extremely fine and the speed of agitation should be 1,200 ft. per minute. The concentrates will average about 50% antimony, and the recovery will reach 90% as a maximum.

To get further light on the subject of concentration of antimony, analysis was made of the paper by Donald G. Campbell, to which reference has already been made. His tests were made on an antimonial silver ore containing pyrite, stibnite and sulphides and antimonides of silver, totaling about 5 or 6%. The oils used were cylinder and machine mineral oils; sperm, fish and whale oils; commercial cottonseed and rapeseed oils; china-

FLOTATION TESTS ON ANTIMONY ORES

	Nome district	Fairbanks district	Antimonial-Silver
Ratio ore to water.....	1 to 3	1 to 5	Not given
Mesh.....	Through 100 on 150 (Tyler standard)	Through 100 on 150 (Tyler standard)	— 100 (?)
Acid.....	H ₂ SO ₄ (sp. gr. 1.84) 0.30 %	H ₂ SO ₄ (sp. gr. 1.84) 0.44 %	H ₂ SO ₄ (sp. gr. 1.84) 0.40 %
Oil.....	Wood creosote 0.20 %	Creosote 1.20 %	Pine tar 0.10 %
Temperature.....	13°C. (55.4°F.)	26°C. (78.8°F.)	18°C. (65°F.) and 65°C. (150°F.)
Time of agitation.....	15 minutes	15 minutes	2 minutes
Peripheral speed impeller..	950-1,320 f.p.m.	1,200 f.p.m.	1,100-1,200 f.p.m.
Per cent. of antimony in concentrates.....	30-54	50-65	Not given
Per cent. of antimony in tailings.....	4-14	8-20	Not given
Percentage recovery.....	40-90	60-90	39-79
Ratio of concentration....	3.5 (approximate)	2.0 (approximate)	Not given

wood oil; resin oil; pine, and pine-tar oils. The final results of the investigation are shown.

No conclusions can be reached from a study of this investigation, but it seems to indicate that the results from the use of pine-tar oil were not as good as those in which creosote oils were employed. A parallel to this behavior may be found in experiments Nos. 48 and 49, in the larger table, in which pine oil was used on the Alaska ore.

The data available on the subject of the flotation of antimony sulphides are not complete enough to warrant any final decision, but the evidence at hand seems to indicate that stibnite can be successfully concentrated with high recovery by the use of sulphuric acid and creosote oil in any of the mechanical-agitation types of flotation machines.

Concentrating Molybdenite.¹—Up to the present the main production of molybdenite in Canada has been benefited by the film-flotation method, either on the Wood machine or by employing modifications of the same principle, such as in the designs developed at the Dominion ore-testing plant at Ottawa and also at the mill at Renfrew, Ont., built by the International Molybdenum Co. Modern methods for the treatment of molybdenite other than film flotation are confined to oil flotation, which, although not yet established in Canada to the same extent, has been and is being applied successfully to molybdenite in different parts of the world. Water concentration can hardly be recognized as an efficient method in itself, although it was formerly used; but the extreme softness of the mineral causes the losses by attrition to be excessive, and the thin flakes and fine mineral float away in the tailings. Neither can dry methods of separating by a system of screening be regarded as efficient except in rare cases.

Character of Canadian Molybdenite Ores.—Most of the molybdenite ores at present of commercial importance in Canada contain a high percentage of iron in the form of pyrite and pyrrhotite, and also mica, but they seem to be sufficiently free from such minerals as bismuth, nickel or wolfram not to require any consideration for the separation and recovery of these minerals, as is sometimes the case in other parts of the world; for example, in Australia. To this extent Canadian molybdenite ores, con-

¹ Abstract of a paper by H. H. Claudet, read before the March, 1917, meeting of the Canadian Mining Institute.

taining only one economic mineral, present a fairly simple problem, and as future practice tends toward the smelting of a much lower grade than formerly, the mill work should be further simplified thereby. It is clear that the efficiency of a concentrator will be much more easily attained if the concentrate required is only 40–50% MoS_2 instead of 90%. There seems to be no scientific reason why good practice in smelting molybdenite should not obtain with the former figure, as the only advantage in having a high-grade product to treat in the electric furnace for the production of ferromolybdenum is that there is less gangue matter present and therefore less slag to be taken care of. But as iron is required as a flux, the presence of pyrite or pyrrhotite, which with the prevailing ores would be the principal concomitant in a lower-grade molybdenite concentrate, would be acceptable up to a point. To offset the disadvantages of extra freight on transportation of concentrates and increased furnace costs when viewing the smelting of a 40%, 50% MoS_2 product as compared with a 90% product, there would be—a simpler design of mill, a more easily operated mill and a higher percentage of values recovered. The balance easily seems to be in favor of the lower-grade concentrate.

The film-flotation system, now extensively in use and well established as an efficient method, requires the ore to be dried in order to slightly oxidize the iron pyrite and other pyrites without affecting the molybdenite. This is rather a disadvantage, as a dry-crushing mill presents undesirable features compared with ordinary wet crushing. This method depends on the surface tension of the water; the dry molybdenite floating on the surface forms a tenacious film, while the gangue and oxidized sulphides are more easily wetted and sink. The usual practice is to crush the ore to 20 or 30 mesh, and this operation on the dry material produces considerable dust which has to be collected and is probably untreated, although it is customary to put this impalpable powder into the flotation machines. It is advisable to use a Wilfley table to retreat the tailings after flotation in order to catch the larger pieces of molybdenite, which sink owing to their being rounded. It is necessary to retreat the products in order to get clean concentrates, but separate units, each having a capacity of about 6 tons per 24 hours, can be arranged conveniently in a mill without taking up much floor space.

FLOTATION TESTS ON STIBNITE ORE FROM THE FAIRBANKS DISTRICT, ALASKA
 Original Ore: Sb, 35.0%; Insol., 41.4%. Case Machine Used

Exp. No.	Ore in Grams	Mesh	Water, c.c.	Oil— c.c. %	Type of oil used	Sulphuric, c.c.	Acid %	Temp. °C.	Time min.	Speed, r.p.m.	Concentrates			Tailings			Ratio concn. % Sb	Remarks
											Wt. %	In- sol. %	% Sb	Wt. %	In- sol. %	% Sb		
1	300	-65	1,200	1.2	0.41	Wood creosote	0.70	0.43	10	1,800	97.10	0.66	7	193.60	0.16	6.67	23.10	Specific gravity wood creosote is 1.024
2	300	-65	1,200	1.2	0.41	Wood creosote	1.05	0.65	10	1,800	86.9	2.37	5	174.58	0.17	2.67	0.3.50	
3	300	-65	1,200	1.8	0.62	Wood creosote	0.40	0.00	10	1,600	133.17	8.55	1	159.60	5.17	6.72	2.25	
4	300	-65	1,200	1.8	0.62	Wood creosote	0.35	0.21	10	1,600	112.10	0.64	8	168.65	2.15	0.72	2.68	
5	300	-65	1,200	1.8	0.41	Wood creosote	1.48	0.86	10	1,800	96.7	8.65	3	183.60	7.16	4.70	8.10	
6	300	-65	1,200	1.2	0.41	Wood creosote	1.38	0.91	13	1,800	93.9	6.1	3	134.62	8.16	2.71	7.3.20	
7	300	-65	1,200	1.2	0.41	Wood creosote	0.35	0.21	10	1,600	135.10	2.38	4	140.65	8.12	7.80	2.22	
8	300	-65	1,200	2.4	0.62	Wood creosote	0.35	0.21	11	1,600	122.12	0.65	0	187.62	4.14	9.75	1.2.46	
9	300	-65	1,200	2.4	0.82	Wood creosote	0.35	0.21	12	1,600	135.15	5.56	3	161.65	2.13	8.73	9.2.46	
10	300	-65	1,200	3.0	1.03	Wood creosote	0.70	0.86	23	1,440	112.11	6.65	0	142.60	11.9	82.0	2.68	
11	300	-65	1,500	1.8	1.23	Wood creosote	0.70	0.86	23	1,440	112.11	6.65	0	142.60	11.9	82.0	2.68	
12	300	-65	1,800	1.8	1.03	Wood creosote	0.70	0.43	28	1,700	101.9	0.67	5	177.59	0.16	4.70	1.2.97	
13	300	-65	2,200	1.8	1.03	Wood creosote	0.70	0.43	28	1,700	94.9	7.65	7	161.56	8.17	2.69	2.3.20	
14	300	-65	1,800	1.8	1.03	Wood creosote	0.70	0.43	22	1,700	114.11	2.64	5	194.59	8.17	4.68	7.2.14	
15	300	-80	1,800	1.8	1.03	Wood creosote	1.20	1.04	20	2,600	128.16	0.53	1	158.39	4.20	2.68	0.2.36	
16	300	-80	1,900	1.8	1.03	Wood creosote	0.90	0.74	23	2,000	144.11	6.51	5	137.64	4.17	7.75	6.2.08	
17	300	-80	1,800	1.8	1.03	Wood creosote	1.20	0.74	23	1,900	131.11	5.51	3	148.59	6.20	6.69	0.2.29	
18	300	-80	1,800	1.8	1.03	Wood creosote	0.70	0.43	22	1,600	114.9	5.55	3	183.57	8.22	6.60	8.2.14	
19	300	-80	1,800	0.8	0.10	Wood creosote	2.00	1.23	22	1,500	118.10	4.54	5	158.58	0.20	4.66	7.2.54	
20	300	-80	1,800	0.2	0.30	Acid sludge			21	1,575	135			297				Specific gravity acid sludge
21	300	-80	1,800	0.8	0.40	Acid sludge			22	1,575	138			39.8	190	55	0.22	2.54
22	300	-80	1,800	0.8	0.50	Acid sludge			25	1,500	138			39	8	142	5.20	7.69
23	300	-80	1,800	1.0	0.60	Acid sludge			27	1,570	113			14	0	173	56	2.23
24	300	-80	1,800	1.2	0.90	Acid sludge			24	1,500	96			53	0	174	52	5.25
25	300	-80	1,800	1.4	0.90	Acid sludge	10 drops		26	1,500	98			56	7	195	52	3.24
26	300	-80	1,800	0.4	0.80	Acid sludge			22	1,550	113			54	0	170	54	8.22
27	300	-80	1,800	3.0	1.20	Acid sludge			23	1,550	132			48	7	151	55	2.23
28	226	+100	1,130	1.2	0.50	Wood creosote	0.50	0.40	28	1,550	34	10.0		183	41.0	24.2		80-mesh material retained on 100
29	354	+150	1,770	1.8	0.50	Wood creosote	0.75	0.40	25	1,550	166	10.2	60.3	172	66.4	18.7	75.5	5.2.13
30	370	+200	1,850	2.2	0.60	Wood creosote	0.85	0.41	26	1,500	222	13.6	59.0	142	78.0	8.2	92.0	1.66
31	522		2,610	3.1	0.60	Wood creosote	1.20	0.40	25	1,500	171	19.7	32.5	324	73.2	10.1	62.8	3.08
																		Tailings from Nos. 1, 2, 3; Sb, 16.9%

FLOTATION TESTS ON STIBNITE ORE FROM THE FAIRBANKS DISTRICT, ALASKA—Continued
Original Ore: Sb., 35.0%; Insol., 41.4%. Case Machine Used

Exp. No.	Ore in grams	Mesh	Water, c.c.	Oil — c.c.	%	Type of oil used	Sulphuric o. o.	Acid %	Temp. °C.	Time min.	Speed r.p.m.	Concentrates	Tailings	Recovery, % Sb.	Ratio conc. Sb.	Remarks
												Wt. % In- sol.	Wt. % In- sol.			
32	514		2,570	3.1	0.60	Wood creosote	1.20	0.40	26	26	1,500	220 34.4 38.5	287 68.0 11.9	71.8	2.33	Tailings from Nos. 25, 26, 27; Sb., 23.2%
33	370	+200	1,850	2.2	0.60	Wood creosote	0.75	0.38	23	23	1,075	221	50.0 130 75.8	9.4	90.0	3.80 Pulp from pebble mill
34	287		1,435	1.1	0.40	Wood creosote	0.50	0.32	24	10	1,200	76	185 73.4	13.9	73.0	2.70 Tailings from No. 32
35	500	-100	2,500	3.0	0.61	Wood creosote	1.17	0.40	28	20	1,300	271	55.2 215 75.1	12.1	84.5	2.30
36	500	-100	2,500	6.0	1.26	Japanese creosote	2.40	0.88	31	30	1,500					Sp. gr. 1.046
37	500	-100	2,500	6.0	1.26	Canada creosote	2.40	0.88	26	20	1,550					Sp. gr. 1.050
38	500	-100	2,500	3.5	0.76	Pine tar	1.20	0.44	29	30	1,460					Sp. gr. 1.090
39	500	-100	2,500	2.7	1.20	Pyroligneous acid	1.20	0.44	35	24	1,400	243	53.5 250 75.5	16.8	75.6	2.00 Sp. gr. 1.297
40	500	-100	2,500	4.0	1.03	Stove oil	1.20	0.44	28	21	1,500	293	52.6 206 77.2	9.8	88.5	2.40 Sp. gr. 0.885
41	500	-100	2,500	4.0	1.31	Japanese creosote	1.20	0.44	25	25	1,500	413	39.6 74 77.6	8.9	96.2	6.80 Sp. gr. 0.859
42	500	-100	2,500	3.0	0.87	Union Flotation Acid sludge	1.20	0.44	25	20	1,500					
43	500	-100	2,500	1.2	0.75	Wood creosote	1.20	0.44	26	20	1,500					
44	500	-100	2,500	1.5	0.79	Acid sludge	1.40	0.45	26	20	1,600	184	59.3 285 62.8	19.3	66.6	1.40 Sp. gr. 0.936
45	500	-100	2,500	4.2	0.94	Gashouse creosote	1.20	0.44	27	20	1,600	205	55.2 274 68.1	16.4	73.2	1.80
46	500	-100	2,500	2.5	0.99	Canada creosote	1.20	0.44	27	22	1,400	318	45.5 158 73.4	11.8	88.6	3.20
47	500	-100	2,500	2.5	0.96	Gashouse creosote	1.20	0.44	28	20	1,400	279	52.6 206 74.3	11.3	86.2	2.40
48	500	-100	2,500	4.0	0.59	Canada creosote	1.20	0.44	28	20	1,550	214	57.3 266 70.1	17.1	73.0	1.90 Sp. gr. 0.930
49	500	-100	2,500	3.1	0.76	Pine oil	1.20	0.44	28	17	1,400	245	54.7 237 69.8	14.8	79.4	2.10
50	500	-100	2,500	4.1	0.63	Canada creosote	1.20	0.44	26	15	1,550	232	57.5 254 70.6	15.1	88.0	2.00
51	500	-100	2,500	1.7	0.99	Pine oil	1.20	0.44	28	15	1,550	225	56.4 240 70.6	15.1	87.9	2.10
52	500	-100	2,500	2.6	0.57	Canada creosote	1.15	0.43	28	20	1,450	251	52.7 213	13.1	82.8	2.30
53	500	-150	2,500	2.8	0.61	Wood creosote	1.15	0.43	28	20	1,500	260	59.0 240	11.0	85.3	1.90
54	500	-150	2,500	3.0	0.61	Wood creosote	1.15	0.43	28	20	1,500	260	59.0 240	11.0	85.3	1.90

The Elmore vacuum process has so far been applied more extensively than any other oil-flotation method to molybdenite ores, has been producing molybdenite in Norway for several years, and in 1916 was installed at a molybdenite property in Canada. No sizing, no drying of the ore and no retreatment is necessary under usual conditions, the preferential flotation being obtained by crushing the ore wet to about 10 or 20 mesh and mixing the pulp with a very small quantity of coal oil or other light oil which coats the molybdenite, but not the other sulphides, and in this way the molybdenite separates from the pyrite, pyrrhotite and also the mica. A settling tank is used to thicken the pulp and to overflow any slimes that may be deleterious, which usually contain between 1% and 2% of the total values. One unit will treat about 35 tons in 24 hours efficiently, and when everything is properly adjusted only two products are made, high-grade concentrates and low-grade tailings. The drawback to this method is that there is more machinery to it and it is of a bulkier nature than the film-flotation plant, the installation requiring a high building; but this concerns the construction work more than the operating, while the advantages as compared with a mill of the film-flotation type are: (1) Wet crushing instead of dry; (2) no furnace and no fuel required for drying the ore; (3) marketable concentrates obtained in one treatment; (4) no sizing.

Canadian Government Film-Flotation Plant.—A description of the film-flotation plant now in use at the government ore-testing plant at Ottawa will illustrate this type, which is a modification of the Wood machine. The capacity per 24 hours is from 15 to 18 tons, or about 100 tons weekly. There are three flotation machines for roughing, and one cleaner; so the capacity of each machine can be rated at 6 tons per 24 hours and the floor space for each machine is about 6×5 ft. The ore is crushed to $1\frac{1}{2}$ in. in a Blake crusher, then elevated to a storage bin and from there fed to rolls set $\frac{1}{2}$ in. apart, and after the rolls, $\frac{1}{20}$ is cut out by an automatic sampler. The ore is then elevated in the drier and thence to the Traylor ball mill. It is said that 30-mesh screens give better results technically, but that the 18 mesh is used for commercial reasons; and as with this size the extraction lies between 80 and 90%, according to the grade of ore, which varies from $1\frac{1}{4}$ to 4% MoS_2 , the work accomplished is satisfactory—the higher the feed the better the extraction. After passing through the ball mill, the ground ore is fed dry to the

flotation machines, which make three products: Rougher concentrates containing about 50% MoS_2 and sometimes higher with rich ores; middlings, which are returned to the drier and ball mill to join the circuit for retreatment; and tailings, to waste, containing about 0.2% MoS_2 . The rougher concentrates are dried and sized on a Keedy sizer into a number of sizes for separate treatment. Usually, all the material coarser than 40 mesh will contain 70% MoS_2 , or better, and therefore is marketable; the material finer than 40 mesh is retreated separately and the resulting tailings returned to the drier and ball mill for retreatment. The flotation machines require attention and adjusting from time to time to secure best results, but probably no more than other concentrating machines, all of which depend for their efficiency on the care and judgment of the attendants. The chief drawback, as already stated, is that the crushing operations must be performed dry, and so the cost of fuel for drying, which would not be excessive in a centrally located plant, might be prohibitive in remote districts with no available fuel near-by and where, as is the case in respect to the molybdenite mines in Norway, transportation conditions are difficult. The tailings are passed over a 30-mesh Callow screen, and the oversize is concentrated on a Wilfley table to catch the coarse pieces of molybdenite which are returned to the ball mill for treatment.

The type of wet-crushing mill, including the Elmore vacuum process, similar to those operating in Norway and to the plant installed by the Renfrew molybdenite mines near Mount St. Patrick, Ont., consists of a Blake crusher to reduce the ore to about $1\frac{1}{2}$ in., delivering into an ore bin. From this bin the ore is fed automatically to a ball mill, then elevated and screened on a separate screen of about 1-mm. opening, the oversize returning to the ball mill. The undersize is sent direct to the Elmore machine, where it first passes through a settling tank the overflow of which carries away some of the slime, and the thickened pulp is mixed with the oil and then separated under the vacuum into the two products, concentrates and tailings. This arrangement gives an extremely simple plant to operate when once all the adjustments are made.

Selective Lead-Zinc Flotation.—The problem of developing a process for the separation of lead and zinc is an important one, a practical description of one solution of which was described

by Luther B. Kirtley, in *Engineering and Mining Journal*, July 14, 1917.

The problem was to take for heads, the slimes that would ordinarily be treated on slimes tables, or perhaps the tailings from such tables, and produce by flotation alone, without the aid of tables, lead concentrates low in zinc and zinc concentrates low in lead. The slimes would assay on the average 6% lead and 8% zinc. The maximum percentage of zinc in the lead concentrates that the lead smelter would allow was 15%, and the minimum percentage of zinc that the zinc smelter would accept in the zinc concentrates was 30%.

In order to get high-grade zinc concentrates, it was thought necessary to remove practically all of the lead before attempting to float the zinc, a conclusion that was borne out by the experiments. This condition divided the problem into two natural parts: (1) To get lead concentrates low in zinc, with tailings as low in lead as possible, and (2) to get zinc concentrates high in zinc and with as low zinc tailings as possible.

From the descriptions of the work at the Missouri School of Mines, it appeared that a good oil for lead is hardwood creosote and that a good example of such an oil is Cleveland Cliffs No. 2. Experiments proved that this is a good oil for lead, but that it brings up considerable zinc too. In order to keep the zinc down, the conditions were varied, and it was found that in an alkaline

RECORD OF TESTS USING HARDWOOD CREOSOTE

	Pb	SiO ₂	Zn	Extraction	
				% Pb	% Zn
C. C. No. 2. No reagent					
Head.....	5.0		7.6		
Concentrates.....	19.3	38.1	12.3	55.4	22.3
Tailings.....	2.6		7.2		
C.C. No. 2. Reagent—H ₂ SO ₄ 2 lb. per ton					
Head.....	7.4		10.4		
Concentrates.....	20.0	32.0	13.8	71.1	37.5
Tailings.....	2.9		8.2		
C.C. No. 2. Reagent—NaOH 2 lb. per ton					
Head.....	6.0		9.2		
Concentrates.....	17.4	37.0	11.5	63.5	29.5
Tailings.....	2.8		8.2		
C.C. No. 2. Reagent—Na ₂ CO ₃ 2 lb. per ton					
Head.....	6.0		7.5		
Concentrates.....	11.4	49.0	7.9	66.3	47.4
Tailings.....	3.1		7.0		

solution the zinc content of the concentrates was fairly low and that sodium carbonate was the best reagent.

In all these tests the dilution of the sample was between 15% and 20% solids, and the temperature was the same as that of the mill water. The oil was added at the rate of 2 lb. per ton of dry slimes in each case.

From this series of tests it was concluded to apply the conditions used in test No. 4 and to go ahead with the problem of bringing up the zinc. After a large number of tests it was found that a fuel oil called Texana No. 7, or No. 8 seemed to film the blende with oil in a satisfactory manner and that Georgia pine turpentine No. 205 (which is a mixture of crude pine oil, crude wood turpentine and "heavy products") worked well as a frother. Pensacola pine turpentine No. 350 (which is a crude pinewood oil) and also General Naval Stores No. 17 (which is a crude hardwood oil) worked well as frothers, but as the sample of the Georgia product was in better shape than the others, I used it in most of the tests.

Problem of Low Tailings.—It was soon found that the hard part of the work with zinc was to get low tailings and that increasing the amount of oil above 2 lb. per ton failed to solve the difficulty. Sulphuric acid not only failed to give low tailings, but also produced dirty concentrates. About this time J. M. Callow described some of his methods, and one of them was the use of small amounts of copper sulphate to secure low zinc tailings. By using it at the rate of $\frac{1}{2}$ lb. per ton, satisfactory tailings were made. Copper sulphate was also tried in the lead run and found to be quite a help in securing low lead tailings if used in amounts under $\frac{1}{4}$ lb. per ton.

In all these tests the iron content of the concentrates was about 8%. The different oils seemed to bring up about the same amount of iron under all the conditions. Sodium sulphide, when present in amounts over 5 lb. per ton brought up the iron at a lively rate, but when present in amounts of about 1 lb. per ton, served to bring up additional zinc. This action is especially noticeable if the sulphide is added after most of the concentrates have come over and the effect of the oil has begun to diminish. These last concentrates have considerably less silica, strangely enough, and are frequently high in grade.

In these tests the conditions were the same as those in the lead tests. The mixture of oils was added at the rate of 2 lb.

per ton. The proportion of the two oils varied between 50–50 and 80–20%, with the fuel oil always in excess.

In making the cleaning tests to raise the grade of the concentrates, it was necessary to add some sodium carbonate to the lead concentrates, but the addition of copper sulphate did more harm than good. Without the copper sulphate the zinc content of the cleaner concentrates was about the same as that of the rougher concentrates, but with it the zinc seemed to come up at the same rate as the lead. In cleaning the zinc concentrates the addition of reagents seemed to be of no help. In all cleaning tests the addition of oil was unnecessary.

In making the tests, both the Minerals Separation type and the Callow type of machine was used. The latter seemed to give lower tailings than the Minerals Separation type, and concentrates that were broken down easily and easily cleaned as

FLOTATION TESTS USING FUEL OIL

	Pb	SiO ₂	Zn	Extraction	
				% Pb	% Zn
Texana No. 8, Georgia No. 205. Reagent—Na ₂ S 2 lb. per ton					
Head.....	2.4		6.0		
Concentrates.....	11.8	24.0	26.4		72.1
Tailings.....	0.3		2.0		
Texana No. 8, Georgia No. 205. Reagent—CuSO ₄ ½ lb. per ton					
Head.....	0.8		4.9		
Concentrates.....	11.8	32.0	20.9		71.0
Tailings.....	0.2		1.7		
Texana No. 8, Georgia No. 205. Reagent—CuSO ₄ ½ lb. per ton, Na ₂ S 2 lb. per ton.					
Head.....	3.0		6.8		
Concentrates.....	12.9	24.0	24.0		81.6
Tailings.....	0.2		1.7		

well, although the silica content was higher than that of the Minerals Separation machine product. The selective action of the oils and reagents seemed to be more pronounced in the Callow type, especially in lead tests.

At one time being short of tubs in which to catch the slimes samples, in lieu of anything better at hand tubs made by cutting an old barrel in half were used. There was enough oil left in the wood of the barrel to introduce some into the sample, and it practically destroyed the selective action. After that, care was taken to use only the galvanized iron tubs for the samples.

During the winter months the oils became cold and stiff and hard to handle. To correct this difficulty the samples of oil were placed on a little iron shelf on a steam radiator. The habit was formed of leaving the oils on the hot plate for longer and longer periods, and finally for perhaps a week at a time. About this time the results were not satisfactory, and the possibility that the constant cooking had removed some of the volatile constituents of the oils was thought of. Fresh samples of oil showed that this was exactly what had happened, for with the fresh oils the results were considerably better.

As there was but one machine of each type, the same machine was used for both the lead and the zinc runs. The lead content of the zinc concentrates seemed to be the same in cases where the lead tailings were very low as where the lead tailings were not so good. Of course the machine was being cleaned out each time between the lead and the zinc runs, but it was only after fairly flushing out with a quantity of water that the results became more consistent.

The samples in these tests weighed between 15 lb. and 20 lb., which gave products too large to dry and weigh direct. In order to figure the extractions, a formula was required involving four variables, corresponding to (1) heads, (2) lead concentrates, (3) zinc concentrates, (4) tailings. Inasmuch as such an expression seemed unprocurable, Gahl's formula was used, as de-

$$E = \frac{c(f-t)}{f(c-t)}$$

scribed in the *Journal* of Apr. 1, 1916. To adapt this formula to the conditions, a sample of the pulp after the lead concentrates had come off was taken, which was called the lead tailings, and enabled me to compute the extraction of the lead. Then by renaming the same sample "zinc heads" and taking a sample after the zinc concentrates had come off, which was zinc tailings, it was possible to figure the zinc extraction.

In order to determine the weight of dry slime in the wet sample the following formula was used:

$$X = \frac{Sp. Gr.}{Sp. Gr. - 1} W - F$$

which is the same as

$$W = \frac{Sp. Gr. - 1}{Sp. Gr.} + F$$

where F is the weight of water in a given flask, W is the weight of the slimes sample in the same flask, and *Sp. Gr.* is the specific gravity of the dry slimes.

The flask used was a 5-lb. acid bottle, which served two purposes—it gave a sample for assay and also gave the weight of the dry slimes in the bottle. By measuring the volume of the flotation machine with reference to that of the acid bottle, it was easy to determine the weight of slime in the machine.

The final results, showing the effect of cleaning tests, as noted in the table, are actual results and are averages of seven roughing tests the products of which were combined to make one cleaning test.

RESULTS OF CLEANING TESTS

	Pb	SiO ₂	Zn
Average head assay.....	7.0		7.8
Average rougher lead concentrates.....	23.8	32.5	12.5
Average rougher zinc concentrates.....	14.4	23.9	24.3
Average rougher zinc tailings.....	0.7		2.8
Cleaner lead concentrates.....	35.8	21.4	12.4
Cleaner zinc concentrates.....	16.2	14.2	29.3
Extractions:			
Rougher zinc tailings.....	6.8%		23.8%
Cleaner lead concentrates.....	58.3%		18.1%
Cleaner zinc concentrates.....	27.1%		43.7%

In the foregoing examples, it will be noticed that the grade of the zinc concentrates has not been raised much. The assay of the cleaner zinc concentrates is the average from several samples taken during the cleaner test, the first of which was about 35% and the last about 27% Zn. By making the zinc cleaner test shorter, the grade of the product can be raised to 33% or 35% without any bad effects.

The effect of circulating the cleaner tailings back to the rougher cell seems to be beneficial rather than harmful. By doing this a very finely divided oil is introduced, which not only helps to get good results, but also cuts down the consumption of fresh oil.

As may be seen in the flow-sheet diagram, there are two separate circuits. This is necessary to the success of the scheme, for if the rougher zinc tailings were circulated back to the head of the lead machine, they would introduce into that machine oils that would bring up a large percentage of zinc and thus

upset the selective action. Not only would the wrong oils be introduced, but also wrong reagents, and thus wrong conditions would obtain, so that probably the resulting concentrates would contain as much zinc as lead, if not more.

The mixer between the lead and zinc machines is necessary, since the heavy fuel oil used as the collector for the zinc requires thorough mixing, even more than the oil for the lead.

From the results secured it can be seen that it is possible to produce by flotation alone salable products from lead and zinc slimes. That such a system would be delicate goes without saying, for even in plants where simple systems are in operation,

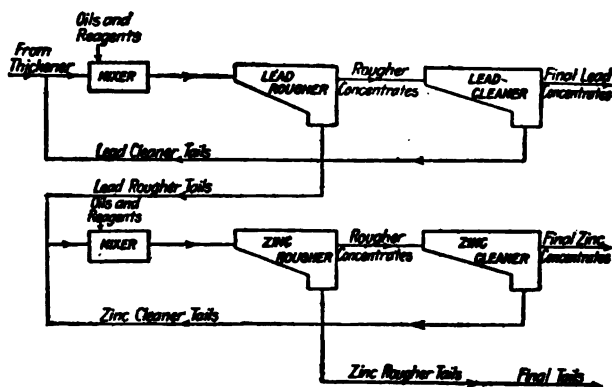


FIG. 51.—Flow sheet of the selective process.

requiring little circulation, conditions vary a great deal at times and occasionally entirely destroy the flotation action. The results cited are not selected in any sense, but are the actual averages of results secured from samples taken from the mill under all operating conditions.

Flotation of a Mexican Silver Ore.—The Real del Monte, Pachuca, Hidalgo, Mexico, has erected a 50-ton Minerals Separation plant at its cyanide plant near Omitlan. A mixture of oils is used, the proportion of each, per ton of ore treated being: Barrett No. 1 coal-tar oil, 1.319 lb.; Barrett No. 609 creosote, 1.319 lb.; No. 80 pine oil, 0.200 lb. Costs of operations are given as \$1.57 (U. S. currency) per ton, the following results having been obtained:

FLOTATION RESULTS AT REAL DEL MONTE

Extraction by difference—63.9% Ag 60% Au.

Tonnage—Dry pulp 1.038 tons per hour.

Dilution—4.91 water to 1 ore.

Conditions of solutions—Traces of CaO.

Oils employed:

Barrett No. 1 coal tar.....	1.319 lb. per ton
Barrett No. 609 creosote.....	1.319 lb. per ton
Pine tar No. 80.....	0.200 lb. per ton
Total oil.....	2.838 lb. per ton

Contents:	Grams Ag	Grams Au
Heads.....	382	1.25
Tails.....	134	0.50
Extraction.....	248	0.75

Assay of Froth from Compartments

	Ag	Au
1-2-3.....	5.909	22.00
4.....	3.961	14.25
5-6.....	1.779	6.00
7-8.....	1.274	4.00
9-10.....	1.964	6.00
11-12.....	2.595	1.25

At Santa Gertrudis, K. & K. and Callow machines have been installed with average recoveries of 72%, although as high as 78% has been obtained. In the early part of 1918, the experiments having been continued, it was reported that flotation up to that time had not succeeded in giving better than 65 to 70% extractions, but the work was being continued with the hope of securing results comparable with those accomplished by cyaniding. The ores are the typical ones of Mexico—argentite in a quartz gangue.

CHAPTER XIV

FLOTATION CONCENTRATION AT ANACONDA

At the plant of the Anaconda Copper Mining Co., near Butte, Mont., one of the most extensive flotation plants in this country has been put into operation. The practice has been adequately described by Frederick Laist and Albert E. Wiggin in the *Bulletin* of the American Institute of Mining Engineers, March, 1916, from which the review in this chapter has been abstracted.

The concentrator at Anaconda, as remodeled for flotation, consists of eight sections, each of 2,000 tons per day capacity, giving a grand total of 15,000 tons per day, allowing for shut-downs, repairs, etc. All sections are alike with the exception of Section 1. In this section, Hancock jigs are used in place of



FIG. 52.—The Anaconda plant near Butte, Mont.

Evans jigs and tube mills are used in place of Hardinge mills. The drawing shows the flow sheet of Sections 2 to 8.

Milling Division.—The ore is fed from the bins to a 2-in. round-hole shaking screen, the oversize going to a 12 by 24-in. Blake crusher. The product from this crusher is delivered to a 2-in. round-hole trommel, the oversize of which is sent to two 8 by 20-in. Blake crushers. The product from these crushers, together with the undersize from the 2-in. screens, is elevated and passed through 1-in. round-hole trommels. The oversize from

this is treated in coarse Harz jigs, making a middling and a concentrate; the undersize is passed through $\frac{3}{8}$ -in. trommels, the oversize being treated in fine Harz jigs making a concentrate and a middling. All sections are alike up to this point. In Section 1, the undersize from the $\frac{3}{8}$ -in. trommel is screened on $1\frac{1}{2}$ by 12-mm. trommels, the undersize going to the Anaconda classifiers and the oversize to the Hancock jigs. The treatment of the products from this point is the same in all sections, except that Section 1 uses tube mills in place of Hardinge mills for grinding, as noted previously. The undersize from the $\frac{3}{8}$ -in. trommel is screened through 4-mm. trommels, the oversize from these going to the double compound Evans jigs and the undersize going to $1\frac{1}{2}$ by 12-mm. trommels. The undersize from these trommels goes to the Anaconda classifiers, the oversize to double compound Evans jigs. The two sets of Evans jigs make a concentrate which goes to the dewatering bins and a middling which is ground for further treatment.

The concentrate from the coarse Harz jigs is dewatered and conveyed to bins. The middling is screened on a dewatering screen, the undersize together with the hutch product from the coarse Harz jigs going to the Evans jigs. The oversize is passed through rolls, 54 by 24 in., and thence back into the system ahead of the 1-in. round-hole trommels. The concentrate from the fine Harz jigs is sent to the bins. The middling is screened through a dewatering screen, the oversize going to 54 by 24-in. rolls and then back into the system ahead of the 1-in. round-hole trommels. The undersize of the dewatering screen together with the hutch discharge of the fine Harz jigs goes to the Evans jigs.

The concentrate from the Evans jigs is dewatered in bins to about 7% moisture, and sent to the smelter. The jig concentrate assays about 15% insoluble and 8% copper. The middling, together with the hutch product is dewatered in tanks and screened through $1\frac{1}{2}$ by 12-mm. trommels, the undersize from which goes to the Anaconda classifiers, the oversize through 54 by 24-in. rolls, and back to the $1\frac{1}{2}$ by 12-mm. trommels.

The spigot product from the Anaconda classifier is treated on 18 Wilfley tables, fitted with Butchart riffing, making a concentrate and a middling. These tables make a concentrate assaying 25% insoluble and a middling assaying 0.9% Cu. The concentrate is sent to the dewatering bins, together with the fine

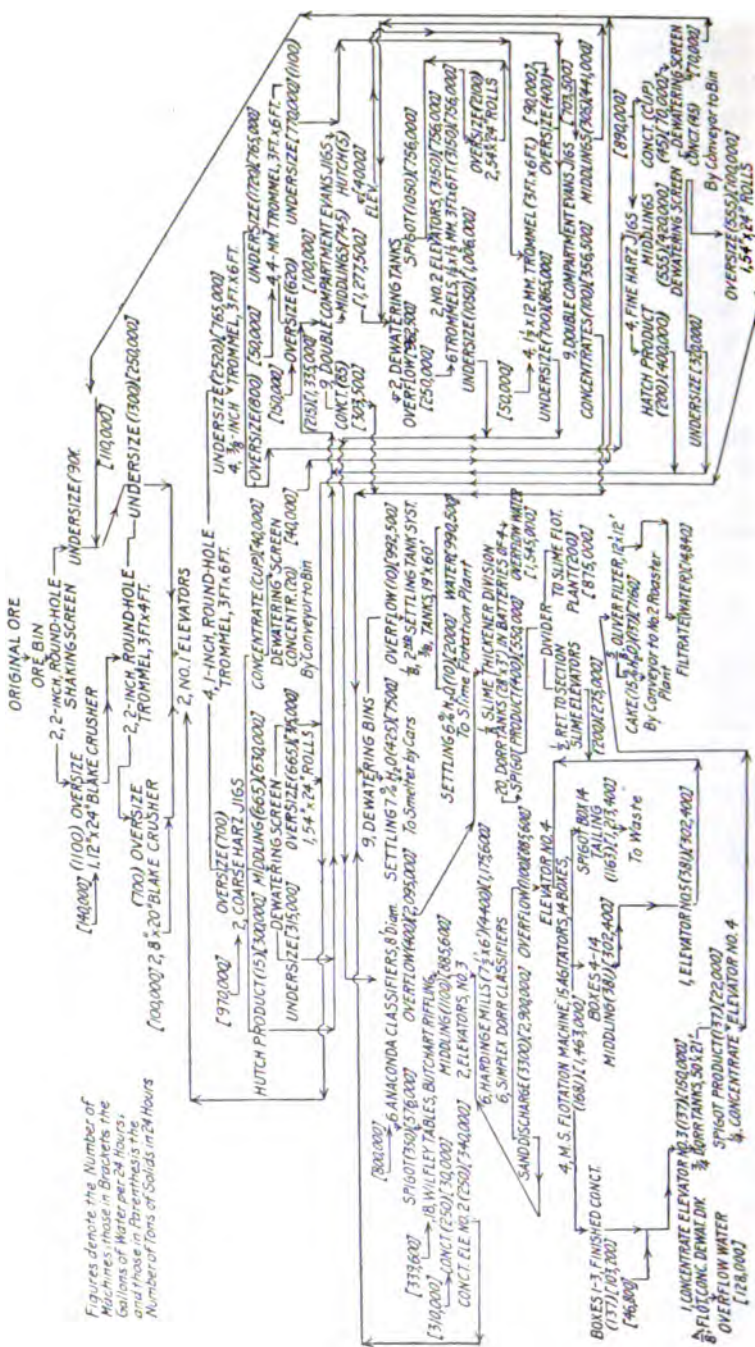


Fig. 53.—Installation of fine-grinding and flotation equipment at the Anaconda concentrator, Sections 2 to 8.

jig concentrate, and the middling is sent to the 10 by 4-ft. Hardinge mills. The overflow from the Anaconda classifiers is sent to the slime thickener division, consisting of 28 by 3-ft. Dorr tanks. The spigot product from these tanks is divided; about one-half is returned to the section and the remainder is sent to the slime plant.

The product from the Hardinge mills is treated in six simplex Dorr classifiers—one classifier to each mill—the overflow going to the flotation division and the classifier sand being returned to the mill.

At the time it was decided to remodel the concentrator, it was not definitely known whether pebbles or steel balls would be used for grinding. To provide for this uncertainty a compromise was effected. The mills were made 10 by 4 ft. and built sufficiently strong for steel balls in case balls were used. Each mill was equipped with a 225-hp. motor directly connected through a flexible coupling. The mill filled with pebbles takes from 95 to 115 hp. to operate. In case steel balls were used it was planned to put in a false wood lining back of the steel lining in the cylindrical part of the mill to reduce the effective diameter of the mill.

This latter plan was finally adopted, and the Hardinge mills will be equipped with the false wood lining, 15 in. thick, in the cylindrical part of the mill, and a Cascade steel lining. With this form of lining, the mill is virtually $7\frac{1}{2}$ by 6 ft. and requires about 225 hp. when loaded with steel balls.

Flotation Division.—The flotation division consists of four Minerals Separation machines each having 15 agitators 3 ft square, and 14 spitzkasten or floating compartments. The agitators for the Minerals Separation machines are of gun metal and are driven by bevel gears from a line shaft, the direction of rotation of the agitators alternating. The machines are made of California red wood; the agitator boxes are further lined with hard maple extending about 18 in. from the bottom of the box. Each machine has an individual drive, power being supplied to the line shaft by a 150-hp. motor running at 385 r. p. m. The speed of the agitators is 225 r. p. m. and as the impellers are 18 in. in diameter, the peripheral speed is about 1,060 ft. per minute.

Each machine makes three products, a concentrate that goes to the dewatering division, a middling that is returned to the

head of the machine, and a tailing that goes to waste. The concentrate is taken from the first three to five spitzkastens and the middling from the last nine to eleven. A portion of the pulp is overflowed from the last three spitzkastens together with the froth. About 6 to 8 lb. of (50°Bé.) H_2SO_4 per ton of flotation feed is used together with 2 to 3 lb. of kerosene sludge acid and $\frac{1}{2}$ to 1 lb. of crude wood creosote. A portion of the wood creosote is added ahead of the Hardinge mill (about 0.03 to 0.05 lb. per ton of feed) and the remainder is added in the sixth agitating compartment. The sulphuric acid and sludge acid are added at the head of the machine. The pulp is heated to from 60° to 70°F., by passing live steam into it at the head



FIG. 54.—Part of the Minerals Separation installation at Anaconda.

of the machine. Three machines are used for treating sand and the fourth for treating current slime from the upper portion of the mill. Each machine has a capacity of about 400 tons per day on sand and 175 tons on slime.

The method of adding the oil and acid is rather unique. The mechanism consists of a revolving disk to which are attached, around the circumference, a number of cups. This disk is set vertically so that its lower edge dips into a pan of acid or oil. As the cups come around they are filled and later discharge their contents into a suitable launder leading to the

flotation machine. The disk is driven by the friction of a wheel against another disk attached to the main drive. The wheel is run at constant speed and by varying the point of contact between wheel and disk any speed desired can be given to the main disk and thus the amount of oil or acid added can be regulated. In addition to the speed regulation, the amount of oil or acid fed may be varied by adding or removing cups or by changing the size of the cups.

In December, 1915, seven sections were operating on the new flow sheet, and the whole mill is now remodeled. The sections were being remodeled one at a time. All the work was done by the company's engineering force. Things had been so well organized and systematized that it required less than 30 days to tear out the old section and install the new equipment, ready for operation.

The following table gives metallurgical data concerning the operation of the mill. Line 1 gives the monthly assay of second-class ore for October, which is the feed to the mill. Line 2 gives the monthly assay of the flotation tailing from the remodeled sections for October. The feed to the flotation machines during the month consisted of reground sand tailing and a portion of the thickened mill slime which was returned to the mill and treated in the fourth machine in each remodeled section. The tailing assay shown below, 0.13% Cu, is for the total sand and slime tailing produced. The sand tailing alone averaged 0.10% Cu and the slime tailing averaged about 0.25% Cu.

METALLURGICAL DATA ON MILL OPERATION

	Per cent. Cu	Per cent. SiO ₂	Per cent. Al ₂ O ₃	Per cent. Fe	Per cent. S	Per cent. CaO	Ounces per ton	
							Ag	Au
Feed to concentrator.....	2.85	59.1	9.4	10.6	11.8	0.6	2.00	0.0070
Total concentrator tailing (flotation).	0.13	81.6	11.3	1.0	0.5	0.5	0.10	0.0005

Following is an estimate of the power consumption per 2,000-ton section, including its proportion of slime treatment.

POWER CONSUMPTION PER SECTION

	Horsepower
Feeding and coarse crushing.....	125
Roll crushing.....	220
Jigging.....	60
Screening.....	37
Conveying and elevating.....	237
Table concentration	25
Fine grinding.....	655
Flotation of sand and slime in mill.....	426
Dewatering mill slime pulp.....	3
Flotation of slime in slime division.....	109
Dewatering flotation concentrate.....	25
Sampling and assaying.....	1
Total.....	1,923
Horsepower per ton of ore.....	0.96

The Slime-Flotation Plant.—Because of lack of space in the mill an auxiliary plant had to be installed to handle the extra slime. This plant consists of 20 Minerals Separation machines of the same type and size as those used in the concentrator, and five 50 by 12-ft. Dorr tanks for dewatering the concentrate. The plant is designed to treat about 2,000 tons of current slime and 1,000 tons of pond slime. This plant was in operation on December 15, 1915. What this plant may be expected to do, so far as the treatment of current slime is concerned, may be judged from the results obtained in the experimental slime machine, which has been operating regularly for more than a year.

The accompanying table gives the analysis of the current slime and the assay of the composite tailing sample for the month of

OPERATION OF EXPERIMENTAL SLIME-FLOTATION PLANT

Per cent.	Feed (current slime)	Tailing	Concentrate
Cu.....	2.10	0.27	12.0
SiO ₂	61.00	67.70	20.0
FeO.....	4.10	2.70	28.0
S.....	4.40	1.10	
Al ₂ O ₃	19.00	18.30	
CaO.....	0.60	0.70	
Ag ¹	1.80	0.20	
Au ¹	0.005	0.001	

¹ Ounces per ton.

June. These are typical of the results which may be expected in the large plant.

The treatment of the pond slime offers a few more difficulties than that of the current slime. This slime has been exposed to

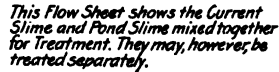


FIG. 55.—Anaconda slime-flotation plant.

the weather for a number of years so that the copper is partially oxidized, consequently that portion probably cannot be floated. Some results from the flotation plant at Great Falls,

covering a week's operation, are tabulated below. This character of material has been treated for some time at the Great Falls plant, and the results should be a good criterion of what may be expected here in the treatment of dump slime.

OPERATION OF SLIME-FLOTATION PLANT AT GREAT FALLS

Recovery based on concentrate.....	78.90
Recovery based on tailing.....	80.90
Average total Cu in tailing.....	0.53
Average soluble Cu in tailing.....	0.13
Average sulphide Cu in tailing.....	0.40
Average total Cu in feed.....	2.52

At Great Falls, one double Minerals Separation machine of the same design and size as used at Anaconda is treating from 300 to 350 tons of dump slime daily.

Sulphidization Experiments at Great Falls.—Experiments covering a period of five days, in which 5 tons of sodium sulphide were consumed, were made at the Great Falls plant, to determine the feasibility of sulphidizing the oxidized copper prior to flotation. Sodium sulphide, in solution, was added at the pug mill together with the dump slime. No difficulty was found in sulphidizing practically all of the copper which went into solution, but it was found that all oxidized copper was not dissolved during its passage through the pug mill and flotation machine.

Moreover, on the addition of sodium sulphide, the froth immediately broke up—a large part of the natural sulphides no longer floating, and the tailing assay showing a marked increase, although the precipitated sulphides floated. Hardly any froth was formed, probably because of the colloidal state of the precipitated sulphides rather than the result of any deleterious action on the oils used of the sodium sulphide, or the sodium sulphate formed.

The cause of the breaking of the froth was not clear. It may have been due to the sodium sulphate formed, or, what seems more probable, to some impurities in the sodium sulphide, the latter being only about 50% pure. This effect was very general, the froth on the second machine breaking up, although no sodium sulphide was added to it, except through a return middling elevator which was common to both machines.

The Flotation-Concentrate Dewatering Plant.—The dewatering of the flotation concentrate is done in six 50 by 12-ft. Dorr

thickeners. Five tanks of the same size have been provided for the slime plant. The pulp is delivered to a baffle box about 5 ft. square in the center of the tank, and extending down to within a few inches of the rake arms. Surrounding this is another baffle about 15 ft. square and extending about 18 in. below the surface of the water. These baffles catch a large portion of the froth which is there broken up by means of a water spray. The capacity of these tanks when treating flotation concentrate is from 200,000 to 250,000 gal. of pulp per 24 hr., although they will probably not be required to handle more than 200,000 gal.

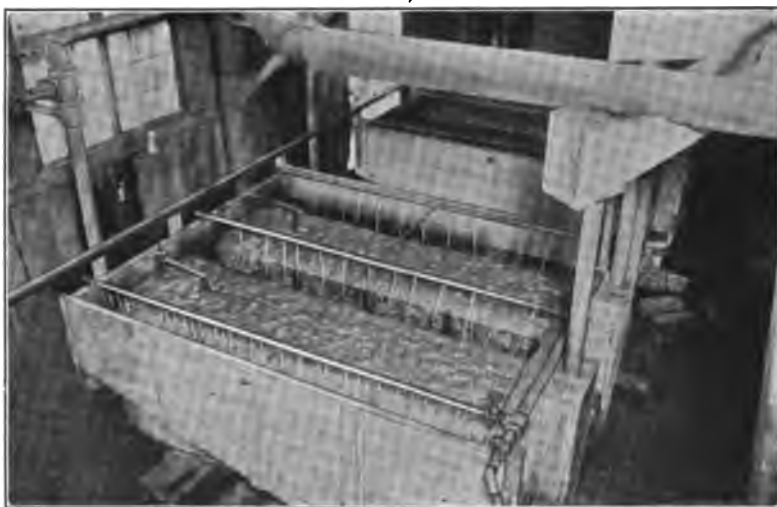


FIG. 56.—A Callow flotation unit in operation at Anaconda.

Operating even at this capacity, there is a small amount of finely divided material that will not settle. Therefore, the overflow from these tanks is run to a slime pond and the solids collected for future treatment. The capacity of these same tanks when treating round-table concentrate is about 1,000,000 gal. of pulp per 24 hours.

Some experiments were made to increase the capacity of the tanks by the use of glue. Indications from tests in a beaker seemed to indicate that glue would greatly aid the settling, but this did not prove to be the case. The glue caused the colloidal material to coagulate, but it did not actually increase the capacity of the settling tanks to any great extent.

The density of the pulp delivered to the tanks averages from 18 to 20% solids, and the spigot averages about 60% solids. The spigot product is further dewatered in Oliver filters 11½ ft. diameter by 12 ft. face. There will be eleven of these, each capable of handling 150 tons per 24 hr. and making a cake containing about 15% moisture. The cake from the filters is delivered to a belt conveyor and is carried with the fine mill concentrate to the new roaster plant. In this way a fairly good mixing of the concentrates will be obtained.

CHAPTER XV

FLOTATION IN PRACTICE

To ores of zinc and lead flotation has appeared to be particularly adapted and, as a matter of fact, even its inventors thought for a long time that its usefulness would be limited to special classes of ores. It was much later that it was proved applicable to copper and other ores. The wide application in Australia was probably due to the existence of so many complex ores in the metallurgy of which some aid was sought. In the United States, too, the first important application was on a zinc-bearing ore—that of the Butte and Superior mine, at Butte, Mont. It has been widely applied, since that time, to other similar ores as well as those of different character.

The Gold Hunter Mill.—The Gold Hunter mill, owned by the Gold Hunter Mining and Smelting Co., is situated in Hunter Gulch, about a mile northeast of Mullan, one of the mining communities of the Coeur d'Alene District. It has a mining capacity of 20,000 tons per year and makes use of concentration by the application of jigs, tables, and oil flotation. The accompanying flow sheet shows in general the operation of the mill. An important use for flotation has been found in treating the Hunter ores profitably, and it may be said that not until flotation became a permanent part of mill treatment was continuously profitable operation possible.

The mill at the present time treats about 310 tons of ore per 24 hours. From this about 35 tons of concentrate is produced, about half of which comes from the flotation department. While under the former system of table and vanner concentration, only about 50 to 60% of the valuable minerals could be recovered, with the addition of the flotation units the total extraction has risen to 80%, thus making it entirely clear what a great difference has been made by the introduction of flotation in this district.

Ore from the mine-ore bin is passed over a conveyor belt to a grizzly having $1\frac{1}{8}$ -in. openings, the oversize of which is

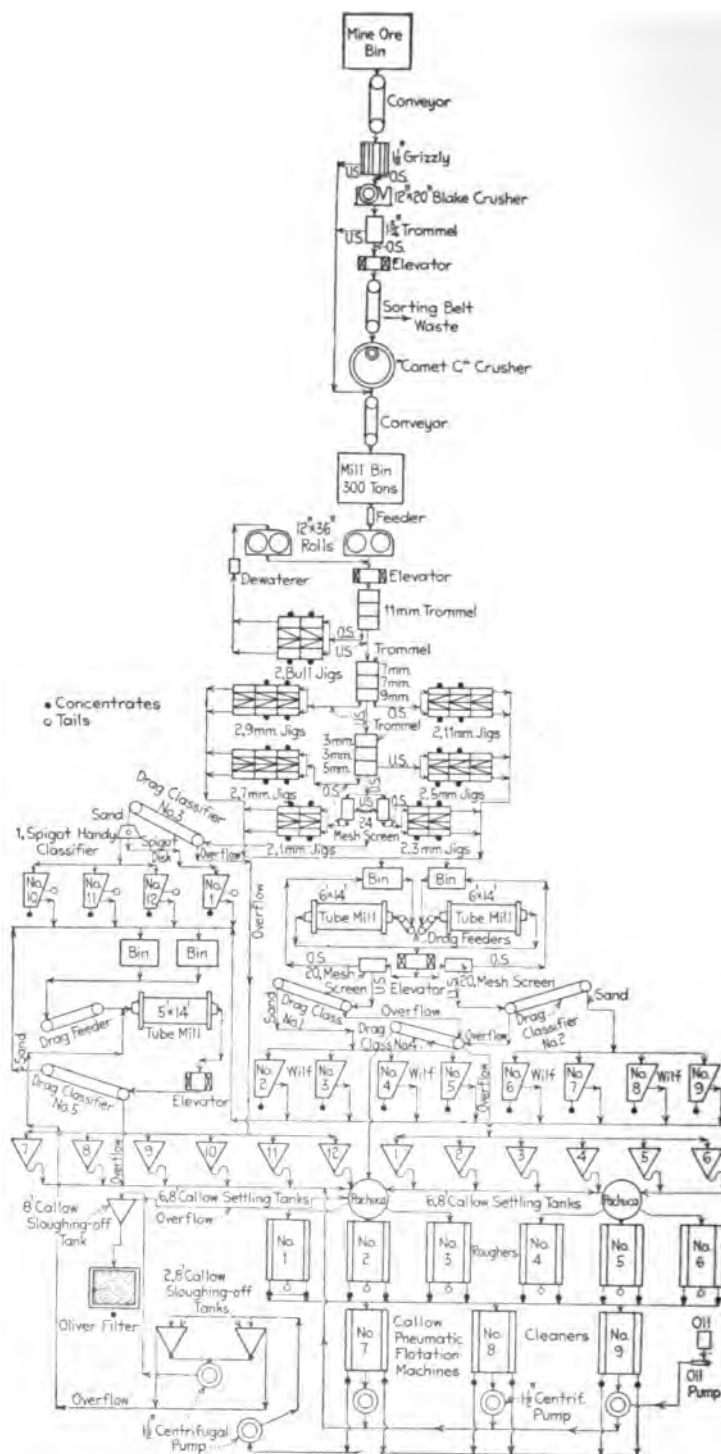


FIG. 57.—Flow-sheet of the Gold Hunter mill, at Mullan, Ida.

put through a 12 × 20-in. Blake crusher. From the crusher the material goes to a trommel with 1¾-in. holes, the oversize being elevated to a sorting belt, where a portion of the waste is removed by hand. The balance of the oversize goes to a Comet crusher, the discharge of which meets the undersize from the trommel and grizzly and is carried on a belt conveyor to a mill bin having a capacity of 300 tons.

From the mill bin the ore is passed through a set of 12 × 36-in. rolls, the discharge from which is elevated to a trommel having 11-mm. openings. The oversize goes to two bull jigs, the tailings of which are dewatered and crushed in another set of 12 × 36-in. rollers, the discharge going back to the elevator which precedes the 11-mm. trommel. From the trommel the undersize is passed to a set of screens and jigs, the flow being clearly shown on the accompanying flow-sheet drawing. The sand is dewatered in a drag classifier and concentrated on four Wilfley tables, the concentrates from which are collected in a bin. The table tailing is then reground in a 5 × 14-ft. tube mill, elevated, dewatered, and reconcentrated. The flow from the jigs is reground, classified, and concentrated on Card tables, as is shown in the flow sheets. At this point preparation is made for treating the tailing from the concentrators by flotation. The pulp is thickened in Callow settling tanks and then delivered into two small Pachuca tanks, where mixture with the frothing oil is accomplished. Creosote oil in the proportion of about ⅓ lb. per ton of ore is mixed in the tanks and then delivered to six Callow roughing flotation machines. Three of these are supplied by each of the Pachuca tanks. From the roughing machines the concentrate is sent to three Callow cleaners, these being machines of the same type as the roughers. Each of them is supplied by two of the rougher machines. The concentrate, finally cleaned, is pumped into two 8-ft. Callow sloughing-off tanks and then to a third Callow cone, whence it is delivered to an Oliver filter. The filter delivers concentrates containing only about 8% of moisture. All solutions from the flotation department are returned to the Pachuca tank for further use, the object being to avoid the loss of any oil that may remain in them.

Among the important points brought out in practice of flotation at the Gold Hunter mill is the necessity of giving much care and watchfulness to the small details of operation. The

flow of pulp to the roughing and cleaning machines must be accurate in quantity, and what is even more important, the dilution must be constant. At this mill the pulp is comparatively thick—that is, contains a high percentage of solids—but as a general rule of flotation this does not apply, since different relations may be necessary for successful working of different kinds of ores. It is important that the proper working conditions, having been determined, shall be maintained absolutely to get best results. At the Hunter mill it takes 5.6 mechanical horsepower to provide compressed air for one Callow cell. It is considered that there would be some advantage in concentrating the flotation tailing on tables and steps may probably be taken to bring about this change.

Silver King and Daly-Judge Mills.—The recent general tendency to remodel concentrating plants so as to include “oil flotation” in the process is aptly illustrated at Park City, where at the old plants of both the Silver King Coalition and Daly-Judge¹ properties, flotation equipment has been installed during the past year. In neither case has the process proved a disappointment and in both improvement has been made over results obtained by the older methods. Not the least interesting feature is the decided difference in the method of applying the process in the two plants.

The presence of a considerable quantity of zinc in the Daly-Judge ore, and the consequent necessity for producing a zinc, as well as a lead-iron concentrate, is largely responsible for this difference. At the Silver King, where the zinc is negligible, the entire bulk of sulphide jig tailings is reground to pass 30 mesh and given a straight flotation treatment, the only recourse to gravity concentration being an occasional separation of the middling concentrate. Even this is necessary only when the ore happens to be unusually high in iron. The old concentrating tables have been entirely supplanted by flotation, and reduced operating and maintenance costs, somewhat better grade of concentrate and increased extraction have been the result.

At the Daly-Judge, on the other hand, only the slime unavoidably made in the usual crushing operations is treated by flotation. The lead and zinc are floated together and then both froth and tailing from the cells are tabled to effect a separation

¹ “Flotation at Silver King and Daly-Judge,” A. B. Parsons, *Salt Lake Min. Rev.*, Feb. 29, 1916.

of the two concentrates. None of the tables have been dispensed with, so that operating and maintenance costs are consequently increased; the advantage derived is an appreciable increase in extraction and a slightly improved grade of zinc concentrate.

Silver King Coalition Operations.—The mine ore is, as heretofore, separated into two portions for milling purposes. The oxidized ore is still treated with jigs, followed by gravity concentrators, flotation being applied only to the tailing from the jigs in the "sulphide" section of the mill. The tailing has been crushed in rolls to pass a $\frac{1}{2}$ -in. trommel aperture and contains on an average of 4 oz. of silver and 1.5% lead. It is fed to two No. 64 $\frac{1}{2}$ Marcy ball mills, 6 ft. in diameter and 4 $\frac{1}{2}$ ft. long, with single-scoop feed. These mills operate with herringbone gear and are directly connected through a flexible coupling to 100-hp. motors, although they consume only about 60 hp. each, operating as at present. The liners are Krupp type, of chrome steel, and each mill is charged with 6,500 lb. of 4-in. chrome-steel balls. This is not the full charge, but is sufficient to handle the present feed. The two mills have ground the $\frac{1}{2}$ -in. tailing to pass 30 mesh at the rate of 600 tons per 24 hr., but are not now at full capacity.

The mills are followed by a revolving trommel equipped with 10-mesh screens, followed by a set of four duplex Callow screens which return the plus 30 mesh to the mills along with the oversize from the trommel. The result is a closed circuit with the undersize from the 30-mesh Callow screens supplying the feed for flotation. The screen analysis of the feed can be governed by regulation of the moisture in the Marcy-mill feed, best results in the cells being obtained when the mill feed is approximately 50% solid. This gives a minimum of very fine slime. The varying character of the ore from day to day makes it necessary to alter conditions frequently, and with different mixtures different sizing has been found preferable. It is the intention to replace the 30-mesh Callow screens with 48 mesh, with a view to meeting more nearly the requirements of all grades of ore, although much of it can be floated satisfactorily with a 30-mesh maximum. A dilution of four to one for the flotation feed is approximated as nearly as possible.

Flotation Equipment.—The actual flotation equipment comprises three units, each including one 3-ft. by 14-ft. Pachuca

mixer, two two-compartment primary-flotation cells, two secondary cells, and one "cleaner." The cells are all of standard Callow pneumatic design. Air for the fifteen cells is supplied at 5-lb. pressure by a Connersville blower handling $8\frac{1}{2}$ cu. ft. of free air per revolution. The blower operates at 210 r.p.m. and is driven by an individual 75-hp. motor.

As originally installed it was planned to use the first four cells in each unit as roughers, in series by pairs; the tailing from the second pair was to be discarded, and the froth from

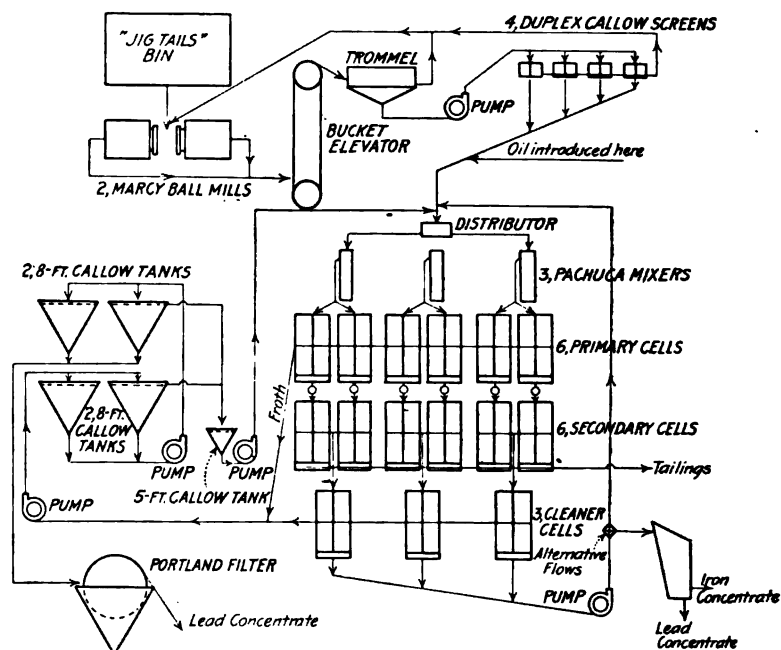


FIG. 58.—Flow sheet of the Silver King Coalition mill, Park City, Utah.

all four was to be cleaned in the fifth cell. The tailing or middling from the final cell was to be returned to the head of the circuit and the froth filtered as the finished product.

However, it was found that a sufficiently high-grade concentrate could easily be floated in the first pair of cells and that it did not pay to put the additional burden on the cleaners. The present scheme, as detailed on the accompanying flow sheet, was accordingly adopted. The transfer from the primary to the secondary cells, which are at the same elevation, is ef-

fectured by six air lifts constructed of 4-in. pipe. Each unit has a capacity of 200 tons per 24 hr., equal to that of the Marcy mills, but as previously mentioned the plant is not being operated at capacity at the present time.

An alternative step in the treatment of the middlings from the cleaner cells is to separate an iron and a lead concentrate on a No. 5 Wilfley table. At times the iron content of the ore is exceptionally high and a representative assay of the middling shows $2\frac{1}{2}\%$ lead, 8% iron, and 6 oz. of silver. Treated on the Wilfley, this gives a high-grade lead concentrate and a marketable iron product running fairly high in silver. This procedure also eliminates to an extent the accumulation of iron that might result from the continual throwing back of the middling into the circuit.

The question of the most efficient and satisfactory oil has been thoroughly investigated and test runs show the best results with a mixture of 20% No. 80. Pensacola crude pine oil and 80% Denver creosote. A satisfactory substitute is an already prepared mixture of No. 1,580 pine oil combined with crude wood creosote. One-fourth pound per ton of ore is allowed to drip through a small pet-cock into the undersize stream from the Callow screens. The resultant froth tends toward brittleness rather than toughness, is rather easily killed, and presents no great difficulty in handling.

Tailing from the secondary cells assays, approximately, 0.5% lead and 1.4 oz. silver, which is an improvement of more than 100% over results obtained in former practice.

It is practically certain that the Pachuca agitators or mixers can be eliminated without any bad effect on flotation. The oil will then, of course, be fed just ahead of the Marcy mills and the action in the mills, as well as in subsequent elevating and screening, would, in all probability, give sufficient mixing of the oil and pulp. This change is to be tried at once.

Handling the Concentrate Froth.—The froth concentrate is run through two pairs of 8-in. Callow cones, in series, preparatory to dewatering in a Portland revolving filter. Two rather novel schemes have been improvised to prevent overflow of the froth in these cones. A sheet-iron ring 10 in. high is put around the inner side of the overflow launder, just inside the rubber belt. This ring is perforated with several rows of $1\frac{1}{16}$ -in. holes around the lower edge, which permit the water to over-

flow but hold back the froth. A large funnel is connected through a flexible $1\frac{1}{2}$ -in. hose connection, to discharge through a pipe running through the side of the tank at a point just about 3 ft. above the apex. The large end of the funnel is adjustable at a desired height above the water line and provides an overflow for any excess foam which would otherwise run over the ring already mentioned. Foam going through the funnel joins the spigot discharge from the cone. The overflow from the four tanks which is not entirely clear goes to a 5-ft. Callow tank, the thickened material from which returns to the flotation circuit. The final dewatering is done in an 8×8 -ft. Portland filter, which gives a rather thin cake carrying as little as 9% moisture. Twenty pounds vacuum is obtained in a small rotary pump of the Connersville type. The filter can handle 75 tons in 24 hr., and with a moderately thick cake it is probable that the resultant product would be as low or lower in moisture.

It is an interesting fact that the filter here seems to handle the heavy galena concentrate with little or no trouble, while at the Daly-Judge an apparently similar concentrate, as will be mentioned later, has so far proved too much for a machine of the same manufacture. The dewatered product is dried over steam coils and shipped.

The advantages of flotation over the former table methods are several, and very clear at the Silver King. The first cost of installation has not been excessive; the operating cost for both labor and power has been decreased; percentage recovery has been increased and the grade of concentrate has been improved upon.

In this connection it is peculiarly interesting to note that, of two concentrates carrying exactly the same percentage of lead, one from gravity concentration and the other from flotation, the latter will assay higher in silver. This has been demonstrated repeatedly, the variation amounting to as much as 10 to 15% of the total silver content. This would seem to indicate a decided tendency toward a selective preference in flotation for silver, and silver-bearing minerals.

The Daly-Judge Plant.—At the Daly-Judge plant, as has been pointed out, the entire situation is somewhat different. Flotation complicates rather than simplifies operations, costs for supplies, labor, and power are increased, and this increase must be more than offset by improved results. The equipment

installed for flotation work includes, besides six standard Cal-low cells, a Dorr thickener, a Pachuca mixer, a small plunger pump for feeding oil, a Portland filter with small compressor and vacuum pump, a Connersville blower and various centrifugal pumps for transferring pulp. In addition there are five Callow tanks and seven Wilfley tables, part of the old equipment, in what may be called the flotation department of the mill.

The thickener is 30 ft. in diameter and 15 ft. deep, constructed with three decks. It takes the slime overflow from a jig and three Richards-Janney classifiers and delivers a slime containing approximately 15% solids, about 90% of which will pass 80 mesh. A thicker product could doubtless be obtained, as only about 35 tons are at the present time being handled in two 8-hr. shifts daily. However, this dilution meets the requirements for floating to best advantage. The pulp is elevated to the head of the flotation department in two stages by centrifugal pumps. The first pump discharges into a mixing box where the oil is added.

The rate of oil feed is regulated by a small plunger pump the speed of which can be altered by means of variable speed pulleys. Further regulation can be had by manipulating a simple lock nut that varies the length of stroke of this pump. The purpose of introducing the oil here is to take advantage of the mixing action of the second centrifugal pump. Even more mixing would probably be beneficial, and if practicable it would no doubt be advantageous to feed the oil at the intake of the first stage pump.

Operations are still being carried out to determine the best oil. Efforts to find an oil which would make possible a preferential flotation of either lead or zinc to the exclusion of the other were fruitless. Various mixtures showed a preference for one sulphide or the other, but it was impossible to get a good extraction of either one without also floating a prohibitive proportion of the other. The only course left was to float as much of both as possible in one operation and rely on concentrators for their separation. Amounts of various oils ranging from 1 to 4 lb. per ton of ore have been tried without a final decision being reached as to the most efficient.

Complicated Variation of Ore Supply.—The problem is somewhat complicated because of the variation in the ore sup-

plied to the mill. Probably the best results, considered from every angle, have been obtained with the Pensacola company's No. 400 flotation oil. This is one of the final residual products of wood distillation, is comparatively cheap, and contains sufficient creosote and oil to give excellent flotation, especially good work being done on the zinc.

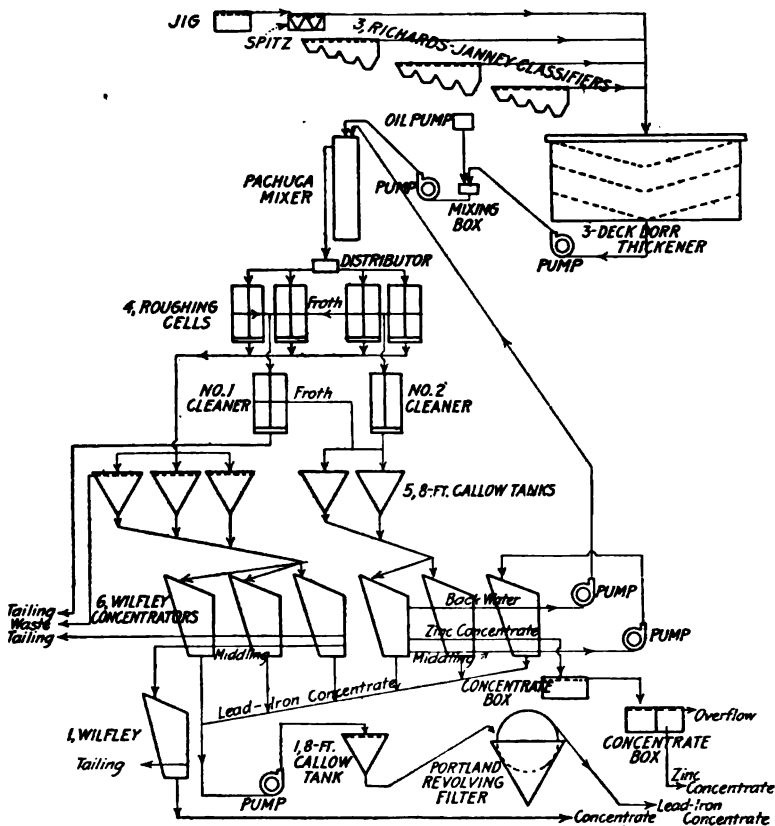


FIG. 59.—Flow sheet of the Daly Judge mill, Park City, Utah.

A single 4×20 -ft. Pachuca tank effects further mixing before the pulp passes to the four roughing cells. The froth is cleaned in No. 1 cleaner, the tails from which are sent to the dump, although it seems unusual that they would not contain sufficient sulphides to warrant further treatment.

The two products, tailings from the roughers and froth from the cleaners, are sent separately to Callow tanks, and are then

tabled on Wilfley concentrators. The tailings are thickened in the Callow tanks, but the froth is simply settled, there being no overflow. Considerable difficulty is experienced in breaking up the froth and this is one of the factors that must be taken into consideration in determining the best kind and amount of oil. Too persistent or voluminous a froth must be avoided. A patent nozzle known as the Koerting spray has been found useful in killing the froth. It is not only comparatively economical in the amount of water required, but at the same time more effective than any other sort of nozzle. The disposal of the various products from the different tables is shown in diagram on the accompanying flow sheet.

A closed circuit is formed in the treatment of the froth concentrate by reverting the "back-water" from the three tables to the head of the flotation circuit. The original flow sheet provided for the return of this product to No. 2 cleaner, from which a concentrate was to be sent direct through the last Callow tank to the filter, and a tailing discharged as waste. This practice threw considerable zinc into the lead concentrate and, while the total metal recovery was somewhat higher, the net result was not so good, especially in view of the high price of zinc.

Following are representative analyses of the two grades of concentrate obtained, as well as of the feed to the roughing cells.

	Silica P.C.	Zinc P.C.	Lead P.C.	Iron P.C.	Silver Oz.
Zinc concentrate.....	3.5	48.0	6.0	4.0	25.0
Lead-iron concentrate.....	1.5	7.0	42.0	16.0	35.0
Flotation heads.....	...	6.0	3.5		5.0

Handling the Concentrate.—The zinc concentrate is collected directly in two settlers, while the lead-iron concentrate is thickened in a Callow tank preliminary to intended dewatering by means of a 3-ft. by 8-ft. revolving-drum filter. The filter thus far has been unable to handle this material to any advantage although the manufacturer still hopes to be able to get results. Obviously the concentrate is extremely heavy, being largely galena, with some pyrite and some zinc. Whether this is the reason for the apparent failure of the filter, or whether there is some fault with the individual machine or installation is not known, but no method has yet been found to keep the concentrate in suspension and prevent its accu-

mulation under and around the drum in a form almost like cement.

There is one feature in which it differs from the Silver King concentrate that may have considerable bearing in the case. At the Daly-Judge the treatment subsequent to flotation would have a tendency to wash off most of the oil coating from the ore particles; while at the former plant, where the froth is directly thickened and filtered, the remaining oil would exert a considerable buoyant force on the solid particles in the filter box. This would doubtless help hold them in suspension until picked up on the drum, and minimize the settling and packing which causes the trouble at the other mill.

Utah Leasing Co.'s Plant.—The new 500-ton plant of the Utah Leasing Co.¹ at Newhouse, Utah, has been built to work over approximately one million tons of tailing from the old Cactus mine. The plant is what might be called strictly a flotation unit, as the entire working up of the tailings is centered in a Minerals Separation flotation machine of newest design. In this respect it differs from most plants in which flotation is used to help in concentration, as at this plant flotation is the sole means of concentrating the ore.

The tailing is brought to the feeding hopper by means of a drag line and from this hopper is elevated to the bins, the elevator being of the continuous bucket type. A conveyor distributes over the ore bins, which have a capacity of about 550 tons. The ore is fed to two 8-ft. by 48-in. Hardinge pebble mills with extra large scoops by means of two Allis-Chalmers Challenge feeders. The Hardinge mills are connected in closed circuit with two Dorr Simplex classifiers, the minus 65-mesh material being sent to an elevator for delivery to the Minerals Separation machine and the oversize being returned to the Hardinge mills for regrinding.

The Minerals Separation Machine.—The Minerals Separation machine is of the type known as "6—6 combination," being 12 cells arranged in series, 6 cells of which are of the single-spitzkasten, 24-in. standard type, and 6 cells being of the double-spitzkasten, subaeration type, known as the Hebbard flotation cell. Eight hundred cubic feet of air per minute at 5-lb. pressure is added to the agitation in the Hebbard cells, the air being supplied by a Roots blower. The entire Minerals

¹ *Salt Lake Min. Review*, Nov. 15, 1915.

Separation machine is driven from a single line shaft, the speed of the impellers being from 425 to 450 r.p.m. The main body of the machine is entirely built of California redwood. The

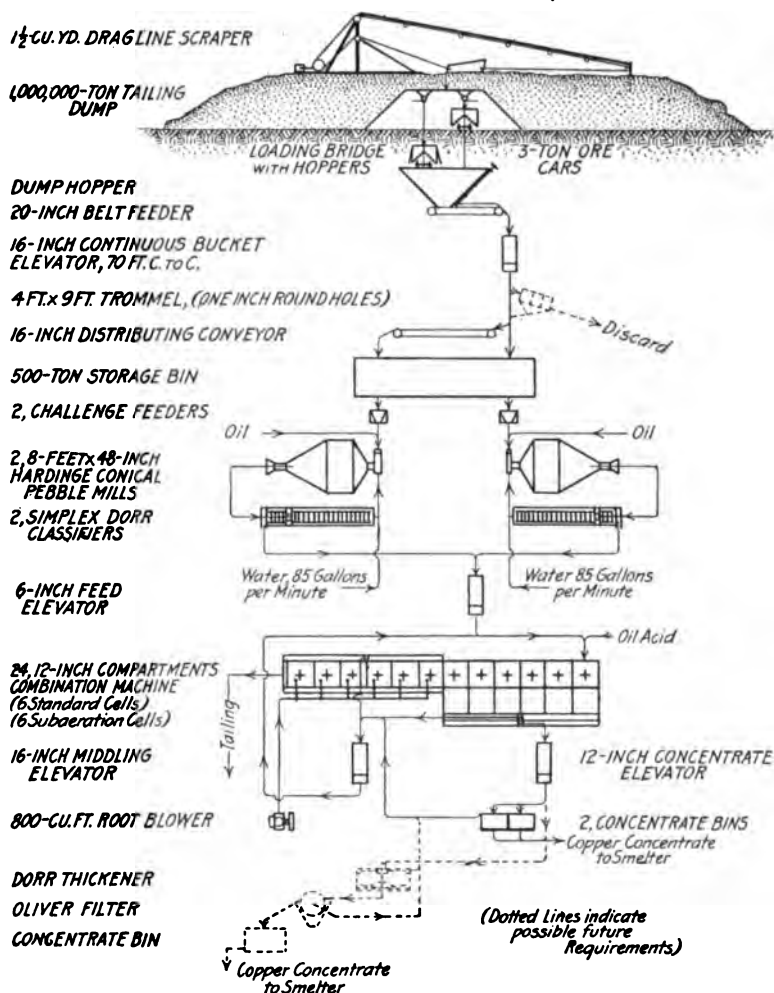


FIG. 60.—Flow sheet of the Utah Leasing Co.'s mill, Salt Lake City.

launders on both the standard and on the subaeration section are of the double type.

Method of Concentration.—A concentrate is made on the first three standard cells, to the first of which the feed from the

classifiers is also added. The subaeration cells are used to make a middling, which is again returned to the head of the machine for recleaning. The concentrates are sent to large bins, with capacity of 200 tons, for settling, and the overflow is pumped back to the Hardinge mills. All launders have been made 14 in. wide and 11 in. deep, the extra size being to take care of excessive amount of froth.

Electric power is used exclusively, the power being received from the Beaver River Power Co., at 44,000 volts, which is stepped down to 2,200 and 440 volts respectively in the mill. The Hardinge mills are driven by a 150-hp. motor, connected to a single line shaft, driving both mills. The Minerals Separation machine is driven by a 75-hp. motor. The drag-line drums are driven by a 35-hp. motor.

The mill is fully equipped with efficient fire protection. Water is secured from the main South Utah pipe line, which has a capacity of 1,200 gal. per minute the source being at Wahwah Springs, approximately 8 miles away. The mill is connected with a storage reservoir, the capacity of which is approximately 90,000 gals.

The Tailing Treated.—The ore in the tailing is almost exclusively a chalcopyrite and is comparatively little tarnished. The copper contents, taking into consideration the slime which averages considerably higher than the sand, is about 0.75% copper with small values in gold and silver. The screen analysis gives the following approximate results:

All through 6 mesh

On 8 mesh.....	5%
On 14 mesh.....	25%
On 28 mesh.....	28%
On 40 mesh.....	18%
On 65 mesh.....	6%
On 100 mesh.....	5%
Minus 100-mesh.....	13%

The best values are in the fine material, the copper contents of the material through 60 mesh being approximately 0.95% copper.

At present it is found that a mixture of light coal-tar creosote combined with kerosene sludge acid has given the best re-

sults and this is what is being used at present. Good results have also been obtained with crude wood creosote and the addition of a small amount of pine oil and sulphuric acid. In general, however, the best results have been obtained in an alkaline solution rather than in an acid solution. Very complete tests indicate that an average extraction of 80 to 85% is entirely feasible. Almost all reagents are added to the Hardinge mills so as to get the benefit of the mixing action. A small part of the reagents are, however, added at the head of the standard cell.

Handling Joplin Slimes.—The treatment of Joplin slimes has been discussed by George Belchic and Glenn L. Allen,¹ who point out that these slimes constitute the material overflowing from the settling tanks and deposits in slime ponds. It may be divided into two classes: (1) slimes formed from a free milling ore, and (2) slimes formed from a "chatty ore," the latter being an ore in which the minerals are finely disseminated in the gangue rock and consequently requires finer crushing. The following screen analyses will suffice to show the difference in fineness of the material, as well as the assay, of slimes formed from chatty and free-milling ores:

Table I.—Slime formed from a chatty ore taken from a Joplin mill. This material assays 6.55% metallic zinc, and 99% of it passes through 150 mesh.

Table II.—Slime made from a free-milling ore, taken from a Galena, Kan., mill, and assaying 7.82% metallic zinc.

TABLE I

	Opening, inches	Mesh	Sample weight, grams	Per cent.	Per cent. cumu- lative weights	Per cent., zinc
All pass through	0.0082	65				
Retained on....	0.0058	100	0.5	0.11	0.11	2.0
Retained on.....	0.0041	150	3.0	0.67	0.78	3.9
Retained on.....	0.0029	200	87.0	19.33	20.11	5.5
Through.....	0.0029	200	359.5	79.89		6.8
Total.....			450.0	100.00		

¹ *Met. and Chem. Eng.*, Nov. 15, 1915.

TABLE II

	Opening, inches	Mash	Sample weight, grams	Per cent.	Per cent. cumu- lative weights	Per cent., sine
All pass through	0.0110	50				
Retained on.....	0.0091	60	9.0	1.84	1.84	3.41
Retained on.....	0.00675	80	5.0	1.04	2.92	2.06
Retained on.....	0.0055	100	12.0	2.48	5.40	2.77
Retained on.....	0.0036	150	21.0	4.35	9.75	3.59
Retained on.....	0.0020	200	268.0	55.48	65.23	7.86
Through.....	0.0020	200	168.0	34.81		10.30
Total.....			483.0	100.00		

Table III.—Gives the analyses of six samples of slime taken from various ponds in Galena Kan., these samples covering the district geographically.

TABLE III

Sample	Per cent., sine	Per cent., lead	Per cent., iron	Per cent., insoluble
1.....	3.77	0.69	4.58	85.87
2.....	7.82	1.09	4.46	70.75
3.....	5.57	0.37	10.78	78.81
4.....	3.69	0.41	3.08	88.87
5.....	7.43	0.45	7.81	81.23
6.....	4.20	0.26	4.18	81.27

The flotation tests were carried out with a laboratory experimental machine, and over a hundred tests were made, in order to establish the following factors:

1. Kind and amount of oil.
2. Effect of adding acid.
3. Effect of temperature.
4. Ratio of dilution of pulp.
5. Speed of agitation of propellers.
6. Time of agitation.

The oils used in the tests were pine oil, creosote oil, tar oil, crude petroleum, crude carbolic acid, cresylic acid, eucalyptus oil, rosin oil, oleic acid, rape seed oil, and various combinations of the above. Tests were also made with mixtures of the above oils with carbon-bisulphide, benzine-sulphonic acid, kerosene,

turpentine, etc. Summarizing the results of the numerous tests the following facts were established:

1. Good results may be obtained without the use of sulphuric acid, although a small quantity (0.5% of the weight of dry slime) in many instances brought the extraction somewhat higher without, however, increasing the zinc tenor of the froth concentrate. With Kansas crude oil the addition of sulphuric acid is detrimental to flotation. Better results may be obtained if benzine-sulphonic acid be used instead of sulphuric acid with both crude petroleum and tar oils.

2. Comparatively high temperature (60°C.) gives better results with light pine oils and rape seed oil, while room temperature (25°C.) is better suited for the heavy tar oils and crude petroleum.

3. Speed of agitation was found best between 1,200 and 1,500 r.p.m. Above 1,500 r.p.m. gangue material is often thrown over with the froth. Excessive speed of rotation produces, in many instances, air bubbles as large as 2 in., which carry readily both gangue and sulphide particles.

4. Pulp dilution varies with the fineness of the slimes. With very fine slimes a 7 : 1 dilution was found to give the best results. With coarser slimes a 3 : 1 dilution worked satisfactory. With those slimes in which a large percentage of the particles is in colloidal form greater dilution is necessary to suspend the particles and allow the oil to surround the sulphide grains. Mechanical adhesion of sulphide and gangue particles in the froth are readily visible in slimes of this character.

5. Time of agitation in these tests ranged from five to ten minutes. This, however, is not so important since on a commercial scale the operation may be continuous.

6. The oils that gave the best results were the following:

Pine oil (specific gravity 0.937) gave a froth concentrate assaying 53% metallic zinc, with an extraction of over 73%; 2.5 lb. of oil and 10 lb. of sulphuric acid were used per ton of dry slime. The slime feed assayed 5.17% metallic zinc with a trace of lead.

Rape seed oil (specific gravity 0.85) gave a froth concentrate assaying 48% metallic zinc, with an extraction a little over 55% when used in quantity of 10 lb. of oil per ton of dry slime, and without the use of acid. The slime feed assayed 5.17% metallic zinc with a trace of lead.

Benzine-sulphonic acid and tar oil (1 : 1 by weight) gave good results when used in the quantity of 8 to 10 lb. of the mixture to the ton of dry slime; a froth concentrate assaying 56% metallic zinc with an extraction of over 50% was obtained. The slime feed assayed 3.69% metallic zinc and 0.41% metallic lead.

Carbon-bisulphide and rosin oil (1 : 1 by weight) gave a froth concentrate assaying 40% metallic zinc, with an extraction of 50% when used in the quantity of 5 lb. of the mixture to the ton of dry slime. The feed assayed the same as the one above.

These preliminary results would indicate that the Joplin-Galena slimes, which at present are a waste product, might be treated on a commercial scale by flotation, and by this means the loss of zinc in milling in this district might be greatly reduced.

CHAPTER XVI

THE PRACTICE OF FLOTATION

The larger copper-mining companies have adopted flotation in many instances with a great measure of success. It has not been so long ago that the principal copper producers were placing a great deal of faith in leaching of the oxide orebodies and considering the roasting and leaching of many other of their low-grade sulphide deposits. All that has been changed, however, and the sulphide ores are now considered to be specifically within the range of flotation. One of the companies that has used the process with success is the Braden Copper Co., operating in the Chilean Andes of South America.

The Braden Mill.—The first Braden mill was built to handle 200 to 250 tons of ore a day and made use of crushers, rolls, Harz jigs, Huntington mills, Overstrom tables and buddles. Extraction was not high, but showed that the ore was amenable to concentration. During the construction of a new mill, the old mill was remodeled for experimental purposes, and in it flotation was tried out.

The ore, a shattered andesite, carrying approximately 75% of the copper content as chalcopyrite, the remainder being mainly bornite and some secondary chalcocite, being brought to the crushing plant from the mine over a 2-km. electric railroad, and after being weighed, was dumped into a coarse ore bin of 2,300 tons capacity. The mill, as now operated, is described by R. E. Douglass and B. T. Colley.¹ From the coarse ore bin the ore, which is usually sufficiently fine to allow 25% of it to pass a 1-in. grizzly, is delivered by steel-belt apron feeders to the 30-in. conveyor belt in two sections serving two gyratories preceded by two inclined grizzlies. One 30-in. magnetic pulley at the discharge of the first section of the conveyor eliminates pieces of iron and steel received with the ore. The undersize of the grizzly goes direct to two 72 × 20-in. rolls and to two 6 × 30-in. trommels.

¹ "Metallurgical Operations at the Braden Copper Co.," *Eng. and Min. Journal*, Feb. 12, 1916.

The 72-in. rolls are run in closed circuits to the trommels, the undersize going direct as a pulp to the concentrating plant proper. A part of the grizzly undersize is at present being sent to a Marcy mill, the first of several that may be installed to increase the grinding capacity.

The discharge of the gyratory crushers, maintained at about a 3-in. maximum diameter, is taken by five 6×30 -mm. trommels, the undersize going to the fine-ore bins of 3,000-ton capacity and the oversize to five 48-in. Symons crushers. The Symons-crusher product of approximately $\frac{3}{8}$ -in. maximum diameter is screened dry, the undersize going to a fine ore bin. The oversize is dropped into four medium rolls, from which the discharge is taken by two 30-in. belt elevators to another set of 6×30 -mm. screens at the top of the crushing plant. From there the oversize is continuously gravitated to the medium rolls and the two sets of 15×36 -in. and two sets of 16×48 -in. finishing rolls running in closed circuits, passing from there to the fine ore bin.

That portion of the ore from the 72-in. rolls running in a wet circuit goes direct to the concentrating plant in a pulp, the dilution being kept as low as possible in order to avoid settling or derangement of the flotation plant from a too-dilute pulp. The pulp is concentrated on the upper tables, of which there are 48 in three sections of 16 each. All are fitted with a modified riffle that permits handling 150 to 200 tons per 24 hr. per table. The concentrate goes by launder direct to four pits at the foot of the mill, and the table-tail screening goes to Hardinge mills. The portion of the ore that was dry coarse-crushed and deposited in the fine ore bins is sent by 3-in. inclined conveyors to three towers feeding that number of sections in the concentrator, each of 1,500-ton capacity. In each section this dry ore falls from the conveyor to eight screens, undersize going to the tables and oversize to two rolls in circuit, from which the undersize to two rolls in circuit, from which the undersize is also sent to the upper tables.

The object of this preliminary coarse concentration is to remove not only clean sulphides, but also the small amount of oxide and metallic copper, often adherent, which, with finer grinding, would be lost in subsequent operations both preceding and during flotation. The feed of these tables is not deslimed, because it is not necessary. Desliming would also add more water to final pulp.

The Marcy mill, recently installed, ground an average of 456 tons per 24 hr.; the screen analysis of the feed and discharge being as follows:

MARCY MILL SCREEN ANALYSIS

	Feed		Discharge
On 38 mm.....	4%	On 10 mesh.....	4%
On 25 mm.....	20%	On 20 mesh.....	16%
On 16 mm.....	24%	On 40 mesh.....	17%
On 8 mm.....	20%	On 60 mesh.....	9%
On 10 mesh.....	25%	On 200 mesh.....	15%
Through 10 mesh.....	7%	Through 200 mesh.....	39%
	100%		100%

The tails of the upper Wilfleys flow directly to five drag classifiers feeding four 36-in. and one 48-in. Hardinge mills per section, a total of 15 mills in all. The overflow of these drags is manipulated to eliminate as much -60 mesh from the drag discharge (mill feed) as possible. The following is a typical performance of a 36-in. Hardinge mill equipped with a britannia lining:

WORK OF HARDINGE MILL

Weight of steel in lining, kg.....	6,900
Weight of cast-iron balls, max. $3\frac{1}{2}$ in., dia., kg.....	3,500
Consumption of balls per ton ground, kg.....	0.75
Horsepower, full load.....	73.50
Tons per 24 hr.....	389
Water-solid ratio of pulp discharged.....	0.52 to 1

SCREEN ANALYSIS

	Feed	Discharge
On 4 mm.....	5%	
On 10 mesh.....	33%	
On 20 mesh.....	37%	30%
On 40 mesh.....	14%	24%
On 60 mesh.....	4%	10%
On 80 mesh.....	1%	5%
On 100 mesh.....	1%	2%
On 150 mesh.....	1%	4%
On 200 mesh.....	1%	3%
Through 200 mesh.....	3%	22%
	100%	100%

On a bench below the upper row of Hardinge mills described is an auxiliary row of nine Hardinge mills, which take the dis-

charge from the upper row through classifiers, as well as the occasional surplus or rushes from the upper Wilfleys that the first row may not be able to handle. The drag classifiers here are manipulated at the overflow in the same way as are the upper mills, in order to eliminate the -60-mesh portion of the pulp as much as possible and to launder it directly to the flotation section. One section of this row and the corresponding section of the upper row are being tried out in closed circuit.

The discharge of the upper and auxiliary Hardinge mills is received by six large double-chain drag classifiers, two in each section, the overflow of which, manipulated to carry as little +60 solids as possible, goes to flotation and the thickened coarse discharge to 42 Wilfley tables, 14 to a section, all equipped with modified riffles.

The tails of these Wilfleys are laundered directly to 18 drag classifiers, feeding the same number of 30-in. Hardinge mills, there being 6 of each to a section. At present these are lined with sillex, flint and local stone blocks; and local and Danish pebbles are used as grinders. Other types of liners and grinders are being experimented with. All these mills are run in closed circuit, with the object of producing the least possible +60-mesh solid in the finished pulp.

The following data are typical of the 30-in. Hardinge finishing mills:

WORK OF THE FINISHING MILLS

SCREEN ANALYSIS		
	Feed	Discharge
On 20 mesh.....	24.3%	
On 40 mesh.....	45.5%	12%
On 60 mesh.....	15.8%	11.2%
On 80 mesh.....	5.3%	8.8%
On 100 mesh.....	3.1%	9.2%
Through 100 mesh.....	6.0%	
On 200 mesh.....		10.4%
Through 200 mesh.....		48.4%
	100%	100%
Ground per 24 hr., tons.....		86.4
Pebble load, kg.....		5,200
Horsepower.....		51.9
Water-solid ratio pulp.....		0.48 to 1

The finished pulp, or overflow from the drags of all sections, is received and sampled in one general distributing box.

The Flotation Treatment.—Until recently all this pulp was treated by flotation in six standard valve-type oil-flotation machines, with one more used for the retreatment of a free-running froth taken from the tail compartments of the six machines. Another plant was used for the retreatment of the slime overflow from the flotation concentrates pit. Four of the plants had 9 frothing compartments, two 11 frothing compartments, and two were of 12 frothing compartments. All were originally fitted with 5-in. circulating pipes. Agitator speed varied from 196 to 214 r.p.m.

Experimental work having demonstrated that increasing the diameter of the circulating pipes to 7 in. and coupling up two plants in tandem would greatly improve extraction and increase capacity, eight plants are being so arranged. The first of these double plants handled 1,460 tons per 24 hr. on the trial run. Experimental work having shown that no ill effects resulted from returning the free-running froth of the end compartments to the head of the machines for retreatment instead of to a separate plant, this practice is being continued.

The reagents used are Swedish or American pine-tar oil, fuel oil, sulphuric acid and a small amount of kerosene to thin the fuel oil. A portion of the pine-tar oil is pre-agitated in the lower Hardinge mills, the balance being added at the head of the flotation machine. The acid and fuel oil-kerosene mixture is added at the same point.

To handle the tails from the Minerals Separation standard plants 28 twin (one common partition) air cells have just been installed. The concentrates from the 28 primary air cells will be retreated in seven additional cells of the same type.

The flotation concentrate is settled in eight pits, four being used alternately. The settled Wilfley and coarser flotation concentrate is loaded directly, by a grab bucket, into side-dump cars for transportation to the acid or smelting plant.

Effect of Flotation on Smelting.—Shortly after the smelting plant began operations, the installation of the oil-flotation process in the mill imposed upon it an entirely new set of conditions. The concentrates produced as shown by this analysis (SiO_2 17.9, Al_2O_3 7.0, Fe 23.0, CaO 0.3, S 30.0, Cu 18.5) greatly improved in grade of copper and of course of sulphur and iron, thus so reducing the silica and alumina that the necessity for limestone as flux disappeared. Also the fine crushing necessary in flotation

methods gave a concentrate in which the alumina or colloidal material was so finely divided that it needed no clay as a binder, but which, on the other hand, was so wet that it could not be briquetted and was so fine that it was most difficult to filter or dry it.

These new conditions entirely changed the details of furnace operation. Where formerly the charge was only fairly fine but highly infusible, it was now all fines and excessively wet, but of much lower silica and alumina content and therefore easily fusible. The coke consumption, which under the first set of conditions had been 12%, now rose to 16% per ton of charge. This increase would not be expected in view of the betterment in fusibility of the concentrate. The reason was that each charge of concentrate went into the furnace in one huge blanket of mud that was absolutely impervious to blast penetration. Two such charges dropped on either side of the furnace successfully plugged all blast passages except the longitudinal center line of the furnace. Here the gases were obliged to escape through an extremely reduced aperture at a greatly increased velocity, carrying with them enormous quantities of heat which, under normal smelting conditions, is absorbed by the charge as the gases pass slowly upward.

During this period, when the flotation process was very new and before the arrival of the filter presses, concentrates of moisture content up to 40%, 50%, and 60% were smelted in this way. There were times when the concentrate would not stay on the belts and when the coke, which was always placed on the bottom of the charge car, would rise through the charge of concentrate and float on top of it.

The blanketing of the charge and the waste of heat through the narrowed aperture, caused by the water as moisture in the concentrate, was found to cost in increased coke a quantity equal to 50% of the moisture content; that is to say, every ton of water added under these conditions required 1,000 lb. of coke, or 50%, to evaporate it.

The 7-ft. ore column was found to be almost too high for such a dense charge, but on the other hand, any effort to reduce the height allowed such large quantities of water so quickly to reach the tuyères that chilling of the whole smelting zone was frequent, resulting in frozen spots or total furnace freeze-ups. The average tonnage per day on such a charge in a 300 × 44-in. fur-

nace was: Concentrate, 135 tons; converter slag, 40; limestone, —; matte, 42; total, 217; coke, 35.

Practice in the Globe-Miami District.—The Globe-Miami district of Arizona is now, and has been for some time, the scene of many important and interesting developments in flotation. The principal mills are operating either on a commercial scale or experimentally, and some profitable commercial results are being obtained. At the Inspiration mill the ore is dumped into a large steel bin. For each of the 18 sections of the mill there is a gate and chute at the bottom of the bin, and underneath the chute a Stephens-Adamson conveyor-type feeder that drops the ore upon an inclined conveyor belt, carrying it up to the feed box. Here half goes to the first and half to the second 8×6 -ft. Marcy ball mill of each section. Two 225-hp. motors operate the ball mills in each section. These mills are set end to end, with two Dorr duplex classifiers between them, one pointed toward the first ball mill and the other toward the second.

From the feeding box the ore enters the first Marcy mill, the flotation oil and mill water being added at the mill feed box. The ground pulp emerging flows around the mill and through one of the Dorr classifiers, where the oversize is pushed up to the feed box of the second Marcy mill, thus completing half the closed circuit. Each Marcy mill in each section receives the coarse from the classifier of the other Marcy mill, together with a fresh supply of ore from the feed box, and the flotation oil and mill water.

The flotation oil is contained in a headless steel drum and is drawn out by a miniature bucket elevator, the bucket being carried upon two continuous chains and running over a toothed head wheel driven by a pawl. The mechanism is similar to some of the usual types of zinc-dust feeders.

The thickness of the undersize from the Dorr classifiers is controlled by a float operating the valves, controlling, through electrically operated mechanism, the flow of water into the slimes compartment. In the seven sections in operation there are three different systems of treatment below the Dorr classifiers.

Difference in Sections at Inspiration.—In sections 1 and 2 the undersize from the Dorr classifiers runs to a set of 12 Callow flotation machines, the concentrate then going to a set of four Callow cells for cleaning. The clean concentrate runs by launder through the concentrate-collecting system, while the tailing is

returned by a centrifugal pump to the first Callow cells for re-treatment.

The tailing in the first group of Callow cells runs to a drag classifier of the Esperanza type. The slime overflows and is combined with flotation oil in a collecting box beneath the classifiers. The sand runs into two hydraulic sizers, each of which has six pockets, and in each pocket there is a pair of Deister baffle-cone classifiers. Fine slime runs into a compartment at the end of the sizers and then to the collecting box under the drag classifiers. Material in this collecting box is elevated by two centrifugal pumps, each carrying half the stream, to a separate group of 12 Callow cells that make clean tailing and concentrate, which go to the same four cleaning cells that clean the concentrate from the first group.

The products from the two pairs of Deister cones making the coarser classification products go together to one Deister double-deck simplex sand table. Each of the other pairs of cones feeds one double-deck table, so that six sizes are made for table treatment; requiring eleven tables per section. The tables make clean concentrate, middling and tailing, but the middling now flows off into the tailing launder. A middling treatment will be undertaken later when the work of completing the mill will permit.

In sections 3 and 4 the flow differs in that the pulp from the Dorr classifiers goes to two sets of flotation machines of eight cells each, while the pumps and collecting boxes return to only one 8-cell machine; otherwise the flow is the same as in sections 1 and 2.

In sections 5, 6, and 7 no Callow machines are used, but Inspiration cells, devised at the property, are at work. The pulp from the Dorr classifiers goes to two machines of 16 cells each, the concentrates from which go to a similar set of two 6-cell machines and make a clean and finished concentrate, the tailing being returned by centrifugal pumps to the feed box of the first group of 16-cell machines. More flotation oil is added at the tailing compartment of the cleaner cells. The tailing from the 16-cell machines flow to a drag classifier, which delivers the coarse material to the Deister sizers, and treatment of the products from there on is the same as has already been outlined. The slime overflows from the drag classifier, together with the slime from the slime compartment of the hydraulic sizers, runs to

waste from the collecting box, instead of being pumped back for further treatment, as is the case in sections 1, 2, 3 and 4.

The main tailing launder leads to three 60×10 -ft. Dorr thickeners, the overflow going to a large sump from which it is pumped back to the new water-storage tanks. The thickened underflow goes to the tailing. The present set of three thickeners is not of sufficient capacity to handle all the tailings, and seven more are to be added—three 100×10 ft., three 80×10 ft., and one 60×10 ft.

Handling the Flotation Concentrate.—All concentrate launders converge to a group of 60×10 -ft. Dorr thickeners, the overflow from which goes to the same sump as the overflow from the tailing thickeners. The thickened underflow is raised by four bucket elevators to 12×12 -ft. Oliver filters, the cake being scraped over a steam-heated plate and the dewatered concentrate dropping to a conveyor belt. It is then elevated to a steel concentrate-storage tank under which the concentrate cars can be started and the concentrate drawn into them for haulage to the smeltery.

The Inspiration flotation machine is built up of 16 compartments, each about 30 in. long by 42 in. wide, the partitions between cells being so arranged that the underflow of the first cell flows successively through all the others, in this respect somewhat resembling the flow through the Minerals Separation machines, but the flow is by gravity, or displacement, there being no agitators to lift the pulp. The cells of the 6-cell cleaner machine are smaller than in the 16-cell rougher. The air pipes are above the machines, and a branch from the air main runs vertically downward into each cell, instead of entering from below, as in the Callow cell.

The Globe-Miami district affords some interesting comparisons now in milling practice, in the procedure followed at the Miami, Inspiration, and Old Dominion mills. In the Old Dominion the ore differs from that treated in the Miami and Inspiration mills. A rougher, or bull jig, makes clean concentrate on a mixed feed, the concentrate being about $\frac{1}{4}$ -in. size. The other jigs, together with a rougher or bull jig, recover the larger part of the concentrate produced. A Minerals Separation machine of 16 compartments treats slime, and some Callow cells have been installed for experiment. Flotation here produces a sort of end product—that is, only a small portion of

the entire mill concentrate. Hardinge mills for regrinding and Symons disk crushers in the coarse-crushing department are used, the only rolls being those recrushing the bull-jig tailing.

At the Miami mill, rolls are used both in the coarse-crushing department and in the mill proper, working on the feed from the mill ore bins and the coarse product from them going to the Hardinge mills. Callow flotation cells are used for the final treatment of slime, elevators raising the material while the flotation oil is added at the launder running to the elevators. A Pachuca agitator was used for emulsifying the oil, but has been found unnecessary. The slime from one section is treated in a group of Callow cells, the concentrate from two sections being cleaned in a group of four similar cells.

At the old Dominion mill the 12 head cells of the Minerals Separation machine make a finished concentrate, the last four cells' overflow product being returned by the elevators to the head end of the machine.

At the Inspiration mill the flotation cells treat all the pulp and float some copper-sulphide grains even up to 40-mesh size, and the larger part of the concentrates are made by flotation. The tables recover a lower grade and work on material that has passed through the first, at least, of the flotation cells. The flow of pulp is direct, there being no regrinding once the material leaves the Dorr classifiers. There are no elevators in the mill except those lifting concentrates, and rolls are eliminated entirely. The arrangement of a mill treating 14,000 tons daily is remarkably compact. The plant probably covers less floor space per ton of capacity than any other mill.

The Bunker Hill & Sullivan Mill.—The operation of the Bunker Hill & Sullivan mill, at Kellogg, Ida., has been described by R. S. Handy.¹ It has concentration equipment with a maximum capacity of 585,000 tons per year. The total horsepower required is 1,500. Bronze ball crushers and disk crushers are used as preliminary breakers. The ratio of concentration is $7\frac{1}{2}$:1. The total mill water used is 4,500 gal. per minute, and 45 men are required for operation.

The mill heads assay 9% lead and 3.5 oz. silver, while the tailings run 1.4% lead and 0.9 oz. silver. The percentage of extraction, therefore, is 84% lead and 78% silver. The accompany-

¹ *Eng. and Min. Journal*, July 1, 1916.

ing flow sheet shows the scheme of operation at the mill. The system is about as it has been in recent years, and since the adoption of flotation no radical changes have been made. There is no mineral-carrying pulp leaving the mill except that which flows through the automatic sampling devices.

The present arrangement of flotation equipment is shown in the flow sheet. The vanners are now run to produce a concentrate above 60% lead; tailings are correspondingly higher than they formerly were and are pumped to the flotation thickener, where they are joined by the dirty portion of the slime-thickener

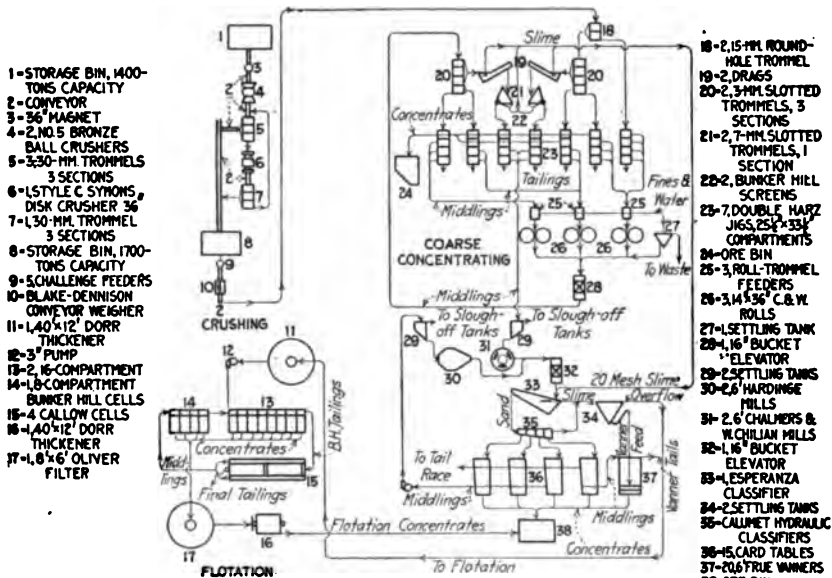


FIG. 61.—Flow sheet of Bunker Hill and Sullivan mill, Kellogg, Ida.

overflows to provide feed for the flotation plant. The details of the Bunker Hill flotation cell are shown in the accompanying drawing. The pumps are a specially made type, with white-iron liners, wearing plate and runner. In the stuffing box is a cast-iron bushing of special design that allows the use of water packing to protect the shaft without unduly diluting the pulp in the cells. The water for this purpose comes from the concentrate thickener and contains considerable oil, which is an advantage.

- Compressed air is blown directly into the suction of these pumps under a constant head, so that this important regulation is easy

a groove is cut so that the cloth cover may be secured by driving rope over the cloth into this groove in the same way that filter bottoms in cyanide tanks are secured.

The grade of froth from the Callow cells is about 30% lead, the object of these machines being to produce low-grade tailings, and the froth is cleaned up in a Bunker Hill machine. The flotation equipment in each section handles from 95 to 100 tons per day of material, about 95% of which passes 200 mesh. There are 16 pumps and 32 cells in the main Bunker Hill section and four pumps and eight cells in the cleaner section. Pine oil is used in one section of the mill and eucalyptus oil in the other. The latter makes a higher-grade froth than the former, and the consumption is a little over half that of pine oil. For keeping the dilution constant at 3 to 1 there is used a Troemner bullion scale, upon the sliding scale of which is laid out a curve giving the dilution due to a given weight of pulp.

CHAPTER XVII

FLOTATION DEVELOPMENT AT THE INSPIRATION MILL

The installation of the flotation process at any mill is not merely a question of mechanical construction. Particularly was this true in the earlier days of the process, when the first step was to make tentative experiments in a process that had made no great reputation—experiments that seemed to be little more than a quasi-scientific amusement. Rarely, however, did such experiments fail to arouse a deeper interest that drew



The Inspiration concentrator.

the experimenter further and further along until, finally, a fine fledged flotation plant, producing commercially profitable results, was the outcome.

The full history of the development of the process at any particular plant is seldom written, and for that reason the history of flotation development at the mill of the Inspiration Consolidated Copper Co., at Miami, Ariz., told by Dr. Rudolf Gahl at the Arizona meeting of the American Institute of Mining Engineers, in September, 1916, is worth recording not only as

an exhibition of what was done at Inspiration, but as a type of what happened at many other plants. In addition, it forms a guide to what ought to be done when a new process is being considered. Dr. Gahl's article is reproduced here from the transactions of the American Institute of Mining Engineers.

The history of flotation in America is very short, at least as far as the large-scale application of the process is concerned. When the Inspiration company first decided to build a concentrating plant, nothing was known about flotation, and the process was to be gravity concentration pure and simple. While plans were being prepared by H. Kenyon Burch (who had been intrusted with the design and construction of the concentrator) the Minerals Separation Co., a concern at that time little known in America, asked and obtained permission to demonstrate the value of its flotation process for Inspiration ore. As a consequence, a small 50-ton flotation machine of standard design was added to the company's test plant and started to operate in the beginning of 1913. This marks the beginning of flotation at Inspiration.

The results obtained with this machine, which was operated by members of the Minerals Separation staff, so far surpassed what this company anticipated that it was decided to continue flotation tests for this purpose, two of the flotation experts of the Minerals Separation Co., I. L. Greninger and G. A. Chapman, were retained. L. R. Wallace, now superintendent of the Miami works of the International Smelting Co., was at that time metallurgist of the Inspiration company and in that capacity took an active part in these tests. Great credit is due to him for his quick recognition of the possibilities of flotation.

Flotation Tests Conducted by Inspiration Co.—The tests led to the conclusion that it would be advisable to incorporate flotation into the concentrating process. Doubt existed only as to the extent to which this should be done. The first design brought out by Mr. Burch only called for flotation treatment of the concentrator tailings. While the tests were progressing, it became more and more evident, however, that flotation should form a more essential part of the milling process, and it was finally decided to leave out all the complications which are usually adopted in gravity concentration plants for the purpose of recovering as much as possible of the mineral values of the fines, and to rely on flotation alone for this purpose.

This decision was reached only after it had been established by numerous tests that the orebody as a whole was suited for flotation treatment. It was found, it is true, that a portion of the ore in the mine was unsuitable for flotation by the process in question, but the amount of this ore was established to be only a small fraction of the total. Besides, the tests proved that the ore contained in this fraction lost its refractory character when mixed with the rest of the ore.

Mr. Greninger gives the following description of the manner in which these tests were carried out:

"Sample lots, each amounting to about 10 tons and representing from 30 to 35 ft., were blasted from the back and sides of the drifts and after suitable crushing and grinding treated in the small flotation machine mentioned above.

"The results were very erratic, some showing good recovery with a high grade of concentrate, while others returned concentrate of a very low grade and also showed a low extraction.

"After a considerable number of tests had been made, it was found that the good results were always obtained when treating ore of a schist gangue, while poor results resulted when the gangue was the altered and kaolinized granite which forms a part of the Joe Bush orebody.

"Ten-ton samples were taken from various parts of the mine for the purpose of determining the extent and amount of this granite ore. These lots were taken from the drifts crossing the orebody from south to north, starting in the granite ore and continuing along each drift until the schist gangue was encountered.

"In all of these tests the results were uniformly good while treating the schist ore and poor while treating the granite ore.

"Subsequently, another series was inaugurated with the view of determining the amount of granite ore that could be mixed with the schist ore without interfering with the treatment of this mixed ore by flotation.

"Various percentages of granite ore were mixed with clean schist and the mixture treated by flotation. It was found that 10% of granite caused no change in the behavior of the ore in the flotation plant and when 20% of the granite was mixed with the schist ore only a slight difference was noted, this difference consisting in a slight lowering of the grade of the concentrate with a corresponding falling off in extraction."

Results Obtained in Small Test Mill.—The test conducted in the small test plant established in a general way the physical conditions under which Inspiration ores could be treated advantageously; for instance, it was decided that raising the temperature did not improve the results obtained in proportion to the extra expense of such procedure.]

So far as flotation oils are concerned, those in charge of the tests came to the conclusion that cresylic acid (98% pure) should be used as the main flotation agent, and should be supplemented by crude turpentine. As the most important result brought out by these tests, I consider the discovery that the flotation agents may profitably be added in the grinding machines, while it had formerly been the customary practice to add the "oils" in agitating tanks especially provided for the purpose. This discovery had an important bearing on the later developments in the Inspiration milling practice, inasmuch as it paved

the way for the use of much heavier oils; for instance, coal tar which it is impossible to amalgamate thoroughly with the pulp in agitating tanks. Mr. Chapman, I think, made this important discovery during this period (U. S. Patent 1102874).

The details of the flow sheet to be followed, were left to be decided by tests on a larger scale. Only this much was settled: That no concentration of any kind, either flotation or gravity, should be attempted before the ore was reduced to the fineness required for the flotation process. This decision was brought about by the fact that excellent recoveries were obtained when this procedure was followed, and was, of course, also strongly urged by considerations of simplicity in the milling process and cheapness of milling operations.

In these tests a 50-ton flotation machine of Minerals Separation Co. standard type was used. In the general design, it was similar to the larger machine later put in use. It had eight agitating compartments with propellers of 12-in. diameter revolving with a peripheral speed of from 1,200 to 1,400 ft. per minute.

Tests in Large Test Mill.—In order to test on a large scale the points already settled in the small test mill, and to decide the points left undecided by the small-scale experiments, a 600-ton test plant was built and put into operation at the beginning of January, 1914. It was my privilege to conduct those tests in coöperation with Mr. Greninger, who represented the Minerals Separation Co., and with the representatives of other concerns, who in the course of time decided to submit flotation machines to the consideration of the Inspiration company. The flow sheet of this 600-ton test mill was extremely simple. The ore after being crushed to the desired fineness by machinery, which was tested out at the same time, was sent to tables; from these the concentrates went to the concentrate bins, while the tailings passed on to an 8-compartment flotation machine of the Standard Minerals Separation type with 24-in. stirrers. Later a 12-compartment machine of the same type was added and operated in parallel with the 8-compartment machine. The tailings from the flotation machines were passed on to other tables, the concentrates from which, combined with the concentrates from the upper tables and the flotation machine, went to the concentrate bins, while the tailings went to waste. The flotation machine was operated in the standard manner, the feed being introduced into the first agitating compartment, passed to the first spitzkasten, drawn to the second agitating compartment, sent from there to the second spitzkasten and drawn to the third agitating compartment, etc., the agitators forming the necessary suction for the transportation of the pulp from the spitzkastens to the following agitating compartments. A final concentrate was made from one or more spitzkastens, while the concentrate from those remaining was considered a middlings product and was returned to the head of the machine for retreatment.

This flow sheet was extremely simple, but after a while even this was considered too complicated, inasmuch as [the necessity of table treatment ahead of flotation treatment was doubted.]

The advantages pointed out in favor of the preliminary table treatment were about as follows: Since the tables would make a certain recovery, an impoverished flotation feed would result and assist the flotation machine to make a tailings product low in copper. However, the validity of this argument was doubted for the following reasons:

In the first place, if complications are to be avoided, the preliminary table treatment has to be of a comparatively rough character; the refinements of hydraulic classification have to be dispensed with, as by its use more water would be introduced into the pulp than would be advisable for the following flotation process. It must be taken into consideration that the pulp delivered to the tables already contains about 3 tons of water to 1 ton of solid matter, experience having shown that the grinding machines, consisting of either ball or pebble mills, deliver a product of the desired fineness, about 1 or 2% on 48 mesh, when the consistency of the overflow from drag-belt classifiers, working in conjunction with the grinding mills was carried at about this figure; experience also showed that this consistency is suitable for the flotation treatment, while greater dilution of the pulp resulted in an increased copper loss. The logical way out of this difficulty would be to introduce settling tanks, for which, however, the modern mill designer has a just abhorrence; at least, when they are to be placed in the middle of the mill. Neither the liberal use of dressing water nor a low tonnage rate would be permissible for the same reason. On this account, a high recovery could not be expected from a preliminary table treatment. During our tests it averaged around 33%.

Furthermore, the assumed fact, that an impoverished feed results in better recovery by the flotation machine, was strongly doubted by some of the flotation experts, who claimed that in order to form a froth which had the necessary carrying power for mineral, it would be better to leave as much mineral as possible in the feed to the flotation machine. In other words, their advice was to leave out the tables in order to permit the flotation machines to do more efficient work.

There is no doubt, however, that tables will catch a certain amount of mineral (especially coarser grains) which will escape in the flotation process. For this reason, tables have to be used to insure the highest recovery,¹ but it seems that tables following the flotation treatment would make up for this deficiency of the flotation machine better than tables ahead, inasmuch as they would not work under the disadvantage, known to every millman, of receiving too rich a feed. It is well known that it is possible to make a much lower table tailing working on feed

¹ This applies only to the flow sheet under discussion.

containing a small percentage of copper than on a feed rich in copper. From theoretical speculations, therefore, no valid reason was advanced that increased recovery should result from table treatment ahead of the flotation treatment.

The next argument advanced by the supporters of a preliminary table treatment was, that it would result in an improvement of the grade of the general concentrates, inasmuch as it would be possible to make a very clean product on the tables, thereby raising the general average of the concentrates. There is evidently some sound foundation for this point; how much, could not well be investigated without resorting to actual experiments.

A third point seemingly in favor of preliminary table treatment is this: Flotation concentrates offer more or less difficulty in mechanical handling; vacuum or pressure filters have to be resorted to for this purpose. It was pointed out, therefore, that if a certain percentage of the total concentrate could be saved on the tables, and a table concentrate produced containing only a small amount of the slime, so troublesome in filter treatment, it might be a decided advantage; the quantity of concentrate to be handled on filters might be materially reduced and economies effected in this manner. At the same time, experience shows that the flotation concentrate resulting under those conditions, because it contains less of the coarser sand, is more difficult for the filters to handle than it would be were the sand left in. Very likely, therefore, no decided improvement in the handling of the concentrates would result from the separation of the table concentrates from the flotation concentrates, at least under the conditions prevailing here.

The same objections do not hold true for tables when used after the flotation process, as in that case no limitations are imposed regarding the hydraulic and dressing water, and the tables cannot help but make an additional small recovery of concentrate. On account of the fact that the table feed is necessarily low in copper, the tailings will be correspondingly low. In all cases, therefore, where flotation treatment does not make a full recovery, or practically so, of mineral values, tabling after flotation seems imperative.¹

Although such theoretical considerations were indulged in, no decision based on them. To settle the main points in question, a series of tests was carried out. On alternate days, the "upper" tables were bypassed while on the remaining days, they were utilized.

Table 1 gives the résumé, on the strength of which it was decided to use secondary tables only and to leave the preliminary table treatment (ahead of the flotation treatment) out of the flow sheet to be adopted in the concentrator under construction. That this conclusion is justified seems to be borne out by the foregoing facts.

¹ This remark again refers only to the flow sheet under discussion.

TABLE 1.—COMPARISON OF EFFICIENCY OF FLOW SHEET No. 1 AND FLOW SHEET No. 2

Description	Flow Sheet	
	No. 2	No. 1
Assay of mill feed.....	1.72	1.67
Assay of flotation machine feed.....	1.72	1.32
Assay of flotation machine tails.....	0.46	0.43
Assay of lower table tails.....	0.29	0.30
Assay of general concentrates.....	32.71	31.72
Recovery upper tables + flotation machine.....		75.30
Recovery flotation machine.....	74.30	
Additional recovery on lower tables.....	9.40	7.40
Total recovery.....	83.70	82.70
Tonnage per deck on lower tables.....	25.70	23.20
Tonnage per deck on upper tables.....		87.00
Oil consumption, cresol + turpentine in pounds per ton..	0.95	0.82

REMARKS.—The recoveries are calculated from the feed and tailings assays and the assay of general concentrates.

CONCLUSIONS.—Roughing on upper tables reduces the tailings assay of the flotation machine slightly, while it does not seem to affect the assay of the lower tables appreciably. Increased recovery results from the use of the lower tables.

In January and February, 1914, the lower table floor was not in operation. For this reason, the flotation tailings were not treated on tables. There was, however, table treatment ahead of flotation, the following results being obtained: Table recovery, 39.42%; flotation recovery, 33.35%; total recovery, 72.77%. From March to August, 1914, various flow sheets were tried; tables were sometimes used preceding and sometimes following flotation, and sometimes in both places. The figures on recovery are, for this reason, not comparable with the preceding ones. From September to December, 1914, no tabs were used preceding flotation; the flotation tailings were, however, treated on tables. The results obtained were: Flotation recovery, 70.83%; table recovery, 7.51%; total recovery, 78.34 %.

Callow Flotation Installation.—While the test mill was in operation, Mr. Callow, President of the General Engineering Co., advised the Inspiration company that he had invented and perfected a new flotation process which, in his opinion, would give as good, if not better, results than the machine of the Minerals Separation Co. As a consequence, arrangements were made to add a unit of machines of the Callow type. The flow sheet contained four rougher flotation cells, which served for the production of a low-grade concentrate, and another cell of the same construction which was supplied for the purpose of cleaning the concen-

trates made on the rougher cells. The tailings resulting from the cleaner cells were returned and mixed with the pulp entering the rougher cells. With the machine was furnished a Pachuca tank for the purpose of mixing flotation oil into the pulp entering the plant, should it be more desirable to do this in addition to or in place of feeding the oil to the grinding machines, as had proven useful both in the small and in the larger test plants. As a matter of fact, the use of the Pachuca tank was abandoned after several months' operation.

The Callow cells work on a different mechanical principle from that underlying the Minerals Separation standard machine, inasmuch as no movable parts are used for the purpose of producing the fine dissemination of air with the pulp, which seems to be essential for any kind of flotation machine; while the Standard Minerals Separation machine depends on the operation of fast-revolving impellers, the dissemination is effected in the Callow machine by blowing air through the pores of a porous blanket, which forms the bottom of each cell.

The Callow machine is ingeniously simple. The feed enters at the upper end of the inclined bottom, while the tailings are discharged through the float valve at the lower end, the concentrates overflowing at the top of the flotation tanks. The porous bottom of the cell is set on an incline to make possible the treatment of feed containing coarse sand. The movement of such sand particles from the feed to the discharge end is thereby accelerated.

The mechanical principle of aeration and agitation by the admission of air through a porous medium, which has been termed "pneumatic flotation," was undoubtedly discovered by Mr. Callow¹ and his associates independently of other inventors, although an earlier patent had been taken out for the same thing in England by Minerals Separation (British Patent, 10929, May 3, 1910) and long before the installation of the Callow machine at this mill, Inspiration ore had been tested in New York by a pneumatic flotation machine constructed jointly by Messrs. Flinn and Towne.² The results of this test were so good that a small test plant of this system was installed at Inspiration.

¹ Mr. Callow refers to his installation at the National Mill, Mullan, Idaho, April, 1914, as the first successful commercial plant utilizing the pneumatic principle.

² This test was made in the presence of Dr. L. D. Ricketts, Consulting Engineer, and W. D. Thornton, Vice-President of the Inspiration company. According to F. B. Flinn, a 600-ton plant of the Flinn-Towne flotation system was shipped to the Tezuitlan Copper Co. in Mexico in the spring of 1913. This could not be put into operation on account of the political conditions prevailing then. Otherwise, this would have been the first commercial pneumatic flotation plant. R. C. Canby informs me that some tests carried out by him suggested the idea to Messrs. Flinn and Towne of constructing a flotation machine to produce the desired fine dissemination of air by blowing air through a porous medium.

Flinn-Towne Installation.—The Flinn-Towne machine, as mentioned, utilizes the pneumatic principle also but the application is somewhat different. An illustration of a single Flinn-Towne machine is given in Fig. 63. The cells are constructed in the shape of cylindrical tanks, the

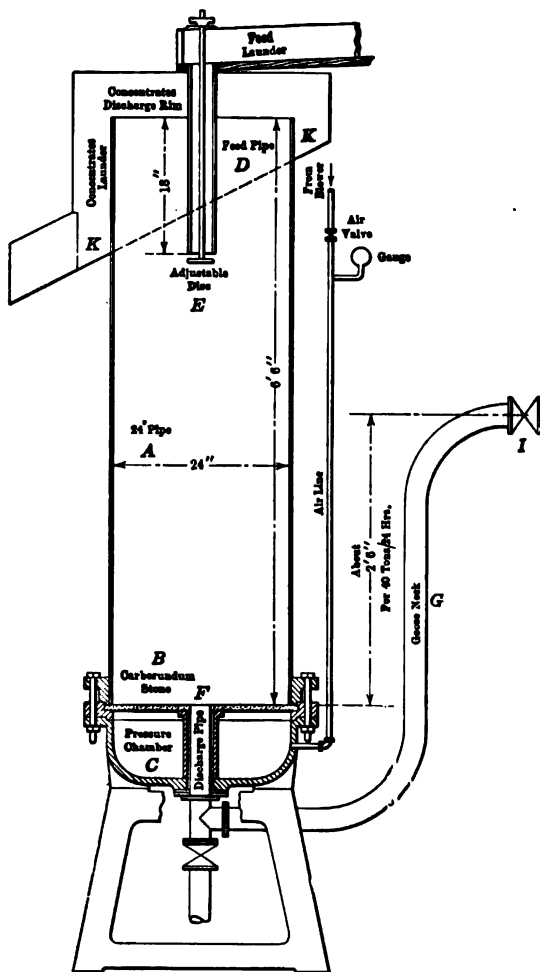


FIG. 63.—Flinn-Towne flotation machine.

bottoms of which are formed by the porous medium. The feed enters near the top in the center of the cylinder, while the tailings leave the machine through a center hole in the porous medium. In the Flinn-Towne installation, only one roughing machine was provided, while additional cells served for the purpose of cleaning both rougher tailings

and rougher concentrates. The tailings produced by the concentrate cleaner cell were returned to the head of the roughing cell. In place of the canvas blanket of the Callow machine, Messrs. Flinn and Towne in the demonstration test at Inspiration used carborundum stones as the porous medium through which the air is injected into the pulp.

The Flinn-Towne machine was withdrawn from the contest after a competitive test between this machine and others had run for several months, although the results obtained looked very encouraging. Its place was taken by a flotation machine designed by Messrs. Cole and Bergman.

The Cole-Bergman Machine.—Their machine is illustrated in Fig. 64. In principle the cells are similar to those of the Flinn-Towne machine. Evidently, the designers tried to improve on this system by mechanically developing the idea underlying the machine, and by changing one point, which in their opinion formed a weak part of the Flinn-Towne machine. This is the construction of the porous diaphragm, which, as explained below, was formed by a round carborundum disk. While carrying on the test of the Flinn-Towne machine, it proved necessary occasionally to wash the carborundum stones by injecting a water pipe from the top and even by removing the stones and cleaning them with water, acids, etc. Messrs. Cole and Bergman ascribed this deficiency to the fact that on account of the horizontal position of the porous medium, sand had a tendency to lodge thereon and to impede or prevent the passage of air through the pores. They substituted, therefore, a system of perforated tubes covered with a suitable fabric, such as canvas or flannel. This led to the construction of a porous medium in the form of a grate, as shown in the illustrations. Their idea seems to be a very happy one, as it makes it possible to apply the flotation process to ore pulps containing comparatively coarse particles of sand; for instance, it seems possible to treat with this machine ore mixtures that contain sand particles as coarse as 10 mesh and perhaps even coarser. Using this machine, the millman is now in a position to treat slime by flotation without the necessity of removing it from admixture with sand. The Cole-Bergman machine has given good results in our tests as have the rest of the machines utilizing the pneumatic principle. It was operated in conjunction with a smaller machine of the same type installed for the purpose of cleaning the concentrates produced on the larger machine.

Methods Followed in Competitive Tests.—In the interest of a fair contest between the flotation machines of the different types, each machine was provided, as nearly as possible, with the same character of pulp as the rest. In the beginning this was accomplished by providing each machine with one or more pebble mills to which feed was sent in such quantities that a mill product of practically the same fineness resulted in each case. When the results of the competitive tests began to approach each other very closely, the equalization of the pulp furnished to

the different machines was still improved upon by mixing the ground product discharged from all the mills, and sending it to a divider which permitted the division into as many parts as there were competing machines, and in any proportion desired. As it had been decided in former experiments not to install tables ahead of the flotation process, no preliminary table treatment was used during the competitive tests, but each flotation system was furnished with a set of tables for the retreatment of the flotation tailings. It developed that this was essential in trying to arrive at a fair valuation of the machines, as some of the machines produced tailings from which additional mineral could be extracted by the table treatment more easily than from the tailings of other flotation machines, the cause evidently being that flotation machines of one type have a tendency to save more of the finer particles, while flotation machines of another type permit of a better recovery of coarser grains. Those flotation machines that make a good recovery on the sand, but leave a larger percentage of mineral in the slime, will not benefit from a subsequent table treatment as much as those of the other kind.

Advantage of Pneumatic Flotation Machines.—In the course of the tests at Inspiration, the fact became apparent that, by the application of the pneumatic principle, a higher recovery of the slime is made possible than without the utilization of injected air, although the Standard Minerals Separation machine also gives excellent results when the tonnage is reduced to a considerably lower figure than the machine is supposed to be able to treat economically.

Therefore, in cases where high power consumption is of little importance, the Standard Minerals Separation machine will fill the requirements of a flotation machine remarkably well. When, however, the power consumed has to be seriously considered, it seems advantageous in the light of the Inspiration experiments to make use of pneumatic flotation for the reason pointed out above, *i.e.*, that poor work of flotation machines on sands can be made up by a subsequent table treatment, while poor work on slime cannot. Within certain limits, it is more important to insist on machines effecting a good slime recovery than on machines making a good sand recovery.

The Minerals Separation Co., realizing this, offered to put in their subaeration type of machine which added the advantages resulting from the use of injected air to the advantages which their Standard machines seemed to have in saving coarser mineral. This resulted in the construction and testing successively of two of their subaeration type machines, one in which the old spitzkastens still were retained, and another one in which they were dispensed with. In these machines provisions were made in the former machine to admit air to the pulp conduits carrying the pulp from the spitzkastens to the agitating compartments next in succession, in order to make this air available to lift mineral to the

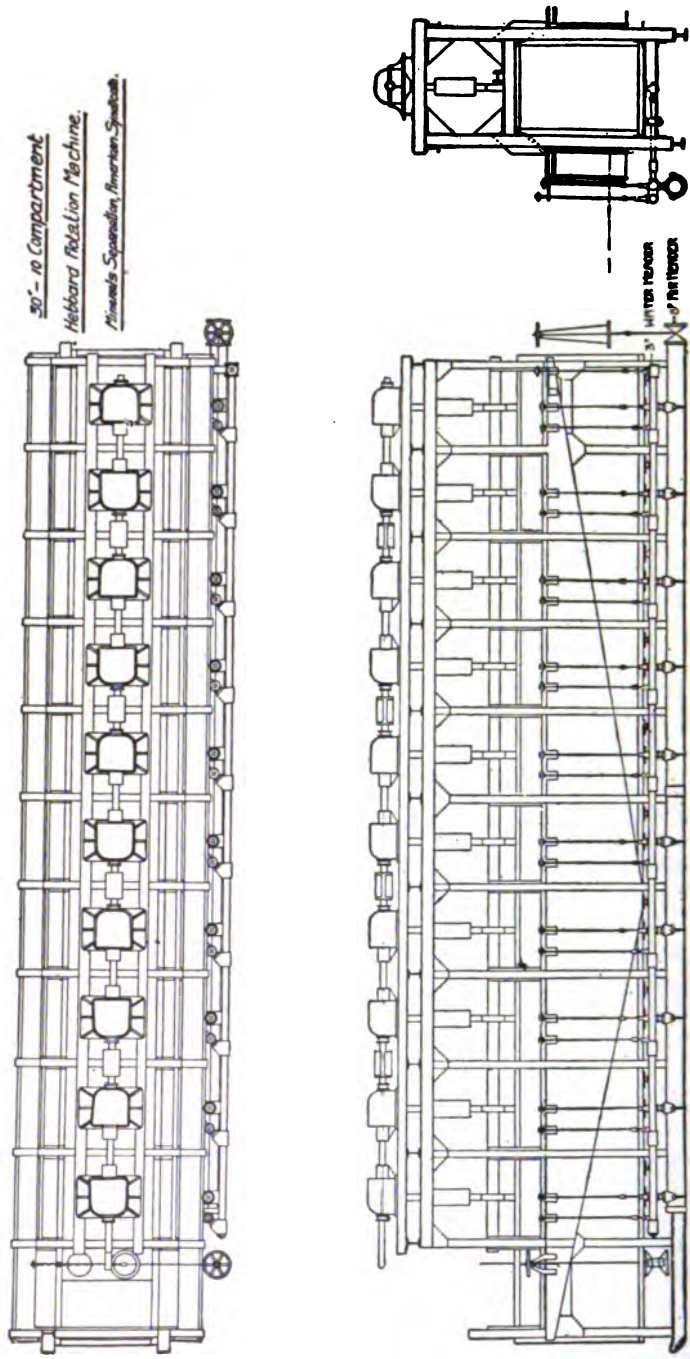


FIG. 65.—Hebhard type of Minerals Separation machine.

top of the spitzkastens where it can be skimmed off. The agitating compartments are closed at the top by covers set on an incline in such a way that the air rises to the top in the agitating compartments and is conducted to the spitzkasten, being thereby utilized for the purpose of carrying the mineral values.

This construction of the Minerals Separation Co. was, however, only an intermediate step toward the design of the second-named machine, a machine of greater simplicity, which is similar in principle to the one illustrated in Fig. 65. The machine consists of a long rectangular tank without any partitions, in which a number of agitators revolve, while air is injected underneath each impeller. We call the machine the "Hebbard Type" Minerals Separation machine.

In order to limit the agitation to the lower part of the machine and to provide an area of comparative quietness in the upper part, a system of baffles is arranged above the agitators. As a consequence, the froth is given a chance to separate out in the upper portion of the rectangular tank, and flows off on both sides lengthwise into launders provided for the purpose.

This machine marks an important progress in the development of the Minerals Separation flotation systems toward increased recovery of copper, at least as far as the Inspiration ores are concerned. In our tests it effected a higher recovery of the fine mineral particles than the old machine had accomplished. The machine also has the advantage of greater simplicity and relatively low power consumption. The only drawback that we have found in the operation of the machine, extending over a number of months, is that the agitating shafts, which are suspended from above, have a tendency to bend and, if not straightened, soon cause the impeller blades to strike against the baffles, breaking one or the other.¹

Float Skimming Device.—While the competitive tests of the different flotation machines were being conducted, it was observed that large quantities of mineral froth frequently appeared in the tailings launders. The idea suggested itself to remove the froth, thus increasing the recovery obtained in the flotation machines and concentrating tables by an additional small amount. After some experimenting, a way was found in which this could be easily accomplished. The tailings launder was widened and deepened for some distance. The tailings stream was forced to pass underneath a baffle on entering the widened space, and was also forced to travel under another baffle board at the end of the space, thus being made to rise through the restricted area formed by the tailings end baffle and the back wall of the float-saving device, which is illustrated in Fig. 66.

¹ This trouble is alleviated in the Hebbard machines turned out more recently by the Minerals Separation Co. by the provision of suitable bearings for the impeller shafts in the bottoms of the machines

Contrary to what might be expected, tailings sand of the fineness of Inspiration tailings will not lodge in the widened-out space unless the length is greater than a certain critical distance, nor will it choke the upward channel in the tailings end of the machine. When conditions demanded the construction of the float-saving device in greater length, it was found that the accumulation of sand, which has a tendency to form in the middle of the machine, can be avoided by arranging an additional

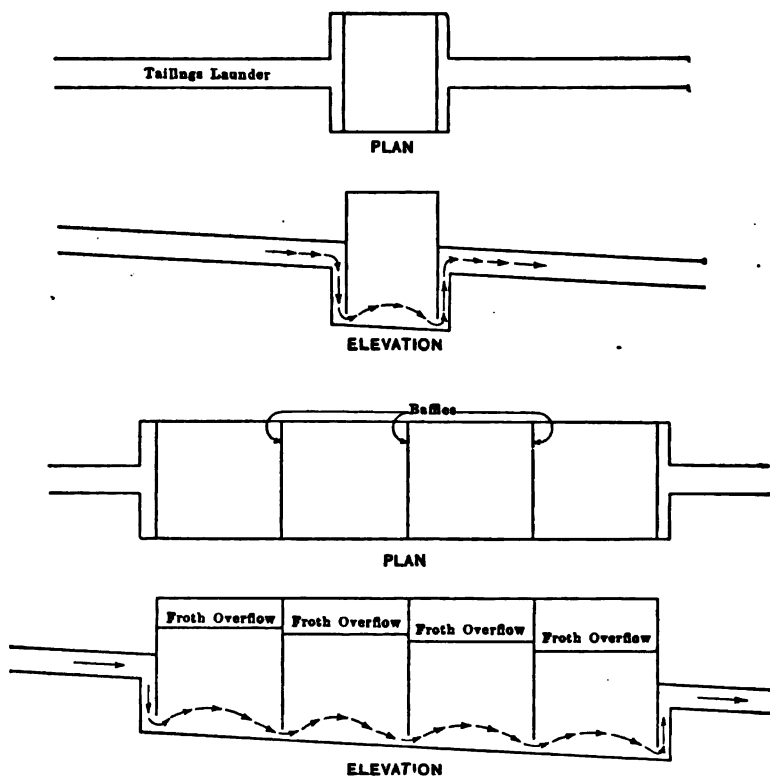


FIG. 66.—Illustration of the origin and development of the Inspiration flotation machine.

baffle not reaching quite to the bottom in the center of the machine. If the machine is built longer yet, more than one intermediate baffle is necessary. The explanation for this behavior of the pulp, which at first glance might seem paradoxical, is that, on account of reduced passage-way between the lower part of the baffles and the bottom of the machine, the pulp is forced to travel through at speed so high that it will not permit the settling out of sand. As a consequence, a hydrostatic head will establish itself between front and back of such a baffle and has to be

taken care of in the design. It is found, however, in actual practice, that the hydrostatic head required to keep the sand from settling out by creating an accelerated current underneath the baffle is relatively small.

A similar consideration explains the fact that the upward tailings passage at the end of the machine has very little, if any, tendency to choke; to take care, however, of possible choke-ups that might be caused by the discharge of coarse rocks, pieces of wood, etc., into the pulp, it was found advisable to arrange some air jets in the bottom of the upward tailings passage to assist in clearing it whenever necessary.

Inspiration Flotation Machine.—This device was in actual operation for several months at our test mill and made itself pay well by adding about 1% to the recovery actually obtained. The question then presented itself as to whether this device, which was successful in saving mineral escaping from the flotation machines could not be transformed into a flotation machine itself by arranging for the injection of finely distributed air into the pulp passing through the machine. The development of the idea led to what we call our experimental Inspiration flotation machine, which was built on a small scale, that is of a size to treat up to 100 tons in 24 hr., and was placed in competition with the other flotation machines already in existence. This machine is illustrated in Fig. 67. The air was injected through a porous bottom arranged in a way similar to the bottoms of the Callow and Flinn-Towne machines (as a matter of fact, a Callow bottom was used in the first tests) and the concentrate flowed over the edge on one side of the flotation tank. The mode of operation followed was also similar to the one developed for the other air machines. A low-grade concentrate was produced and re-treated on a similar machine of the same construction, called the cleaner machine, the tailings from which were returned to the head of the roughing flotation machine. As will be seen from the drawing, the construction of this machine is extremely simple and comes pretty near to an ideal of Mr. Mills, our general manager, who prophesied that the flotation machine of the future would be nothing but a launder with provisions for injection of air. The method of effecting the concentration in two stages was also followed when the new types of the Minerals Separation machines were put into commission.

Arrangement of Callow Cells in Series.—In principle, the Inspiration machine shows one difference from the Callow machine; viz., that instead of splitting the pulp between a great number of machines, it is forced to travel through a series of compartments in succession. This is the same policy that had also been followed in the construction of the Minerals Separation machines. The question then came up, whether it might not be of advantage to arrange the Callow cells also in series. To test out the idea, the flow sheet of the Callow plant was changed. The primary feed was sent to two of the cells, the other cells receiving the tailings from these primary cells.

This arrangement showed at least no disadvantage over the multiple treatment; it seemed superior to the other system in the one respect that it showed at a glance whether the cells were operated well, because in that case most of the concentrate would be produced on the primary cells while the secondary cells would produce a rather light-colored froth and would add only a small fraction to the recovery made on the primary cells. Both primary and secondary concentrates were treated jointly in a cleaner cell, while the tailings from the secondary cells were conveyed to a number of concentrating tables for retreatment.

A variation of the latter flow sheet was also tested. Between the primary and secondary cells, a cone classifier was interposed for the purpose of making a separation between the sand and slime. The slime was sent to the secondary flotation machines, while the sand was passed to tables. This latter flow sheet was finally decided upon for the Callow installation in the large concentrator. Tests made with the object of establishing which of the two is superior, resulted in a tie. Some of the figures on this point are given in Table 2.

TABLE 2.—RETREATMENT OF TAILINGS FROM TWO CALLOW CELLS
SCREEN ANALYSIS OF FEED AND TAILS OF SECONDARY CALLOW CELLS

Mesh	System 1								System 2									
	Feed				Tails				Feed				Tails					
	Per cent. weight		Per cent. Cu	Cu-contents	Per cent weight		Per cent. Cu	Cu-contents	Per cent. sol. Cu	Per cent. weight		Per cent. Cu	Cu-contents	Per cent. weight		Per cent. Cu	Cu-contents	
	Cum.	In-div.			Cum.	In-div.				Cum.	In-div.			Cum.	In-div.			
65	5.0	5.0	0.29	0.014	5.0	5.0	0.21	0.011	0.05									
100	19.0	14.0	0.25	0.035	19.2	14.2	0.15	0.021	0.06	3.8	3.8	0.20	0.007		4.0	4.0	0.17	0.007
150	32.2	13.2	0.29	0.039	32.4	13.2	0.16	0.021	0.10	10.6	6.8	0.25	0.017	10.8	6.8	0.14	0.010	
200	42.4	10.2	0.33	0.034	42.8	10.4	0.21	0.022	0.11	18.6	8.0	0.30	0.024	19.0	8.2	0.25	0.021	
...	100.0	57.6	0.35	0.201	100.0	57.2	0.27	0.155	0.20	100.0	81.4	0.40	0.326	100.0	81.0	0.30	0.242	
Assay calc....			0.32	0.323			0.23	0.230	0.15			0.374				0.28	0.280	
Assay direct...			0.32				0.23		0.15			0.370				0.26		

NOTE.—The concentrates from the flotation cells were in both cases retreated in half-size Callow cells, the retreatment tailings being returned to the head of the primary cells.

	Ton per primary cells
Tonnage rate in system 1.....	49.3
Tonnage rate in system 2.....	47.5

CONCLUSIONS.—No advantage is apparent in favor of either system. System to be installed should be decided from mechanical points.

Variations of Flow Sheet.—In the way of other flow sheet variations, etc., only one thing was tried to any extent; that is, whether it is wise

to send the cleaner tailings back to the head of the rougher machine for retreatment. In the limited time at our disposal, it proved impossible to substitute something better, but the chances are that the same holds true of flotation that is true for water concentration; viz., that middlings are sent back for retreatment mainly because the designer does not know anything better to do with them. In principle it seems wrong to follow this practice, because, when heavy copper losses occur in concentrating machines which express themselves in high tailings, there is always a doubt as to whether this is due to poor recovery made in the primary treatment or to heavy losses incurred by middlings sent back for retreatment.

Coal Tar as Flotation Agent.—It was to be expected, when the flotation process was installed in our test plant, that there would be ups and downs in the recovery because the process was rather new, especially in its application to chalcocite ores. For the month of June, 1914, the recovery obtained in the test mill showed a sudden drop, and the serious problem confronted us of establishing the cause and finding a remedy. Some of the flotation experts suggested that it might be due to the fact that a new shipment of cresylic acid which had been received and used, at that time might not fill the specifications of being 98 % pure. We did not feel competent to say whether the impurities actually amounted to more than 2 %. We were, however, inclined to think that perhaps cresylic acid, which is, as is well known, one of the products resulting from fractional distillation of coal tar, might not represent the fraction most suitable for the flotation of our ores. Having no coal tar available in the neighborhood, we proceeded to make some by distilling a sample of ordinary New Mexico soft coal and separating the tar thus formed into the fractions distilling off at different temperatures.

Our facilities for testing oils were limited at that time. The Minerals Separation Co.'s representative did not believe in small-scale tests, and for this reason did not recommend experiments with small testing machines. Nevertheless, it seemed desirable to have something with which to carry out small-scale laboratory experiments. Dr. Ricketts, who was aware of our troubles and realized the importance of such tests, was kind enough to send us a little electrically operated emulsifying machine, which served admirably for qualitative tests. We also built a testing machine based on the principle of the standard Minerals Separation machine, with the difference, however, that instead of sending the pulp from one agitating compartment to a spitzkasten and then into another agitating compartment and spitzkasten, we made the pulp return from the first spitzkasten to the original agitator, forcing it to revolve in a closed circuit.

The chemist who conducted these tests (Mueller) hit on the idea that it might be well, in addition to trying the different fractions, also to test the coal tar as a whole. The results were very surprising, since they

showed that by the addition of crude coal tar we could effect a greater recovery than we were able to obtain by the use of highly refined cresylic acid. From this point dates our experience that it is better to use coal tar than soluble flotation agents like cresylic acid to save coarse mineral. Cresylic acid is an extraordinarily good agent for producing froth, but the froth which it produces does not seem to have as much carrying power for coarse mineral as that produced by coal tar. Not all coal tars are equally good for this purpose. Tests in laboratory machines easily show the difference between coal tars of different origin.

The system that we used to carry out such tests has been described, although without authorization, by William A. Mueller, one of my former assistants who took part in these laboratory tests.¹

It is difficult to utilize coal tar in plants using flotation supplementary to gravity concentration, on account of the fact that it is not easy to effect a good amalgamation of tar with the pulp in agitating tanks, and even in mechanical flotation machines. The use of coal tar lends itself very well indeed to the system of feeding tar into the grinding machines, a system that, as mentioned above, had been worked out in our small test mill and patented by Mr. Chapman.

The company is indebted to Mr. Callow for proving the merits of coal-tar creosote as a flotation agent by using it in his demonstration plant at Inspiration. After we had established the value of coal tar by laboratory tests, and while efforts were being made to obtain it commercially, he applied creosote successfully. We have continued to use it for a long time, mostly in combination with coal tar, and have only recently dropped it, as we find crude coal tar cheaper and better.

Experience with Primary Slime.—After the first difficulty that we encountered in our large-scale tests had been solved by the introduction of coal tar into the flotation-oil mixture so far used, things went along fairly well for some time until a new difficulty was encountered. It happened that once in a while an abundant froth was produced on the flotation machines, but this froth seemed to have very little carrying power for the mineral contents of the ore and held mainly finely divided gangue. It was observed that this phenomenon occurred with special severity whenever the ores shipped to the test mill contained a large amount of fines and a small amount of coarse rock. It was attributed, therefore, to the presence of what may be called primary slime, *i.e.*, slime not formed by the crushing of ore in the mill, but originating from the mine. That the falling down of the flotation machines was caused by variation of the ores was proven by the fact that samples of the refractory ores when treated in the laboratory testing machine, gave as unsatisfactory results as the corresponding ore did in the big flotation

¹ William A. Mueller: Use of Coal Tar in Flotation, *Engineering and Mining Journal*, vol. 100, No. 15, p. 591 (Oct. 9, 1915).

machines. To demonstrate that the presence of the original slime caused the trouble, samples of the mill feed were separated into sand and by screening them on a 200-mesh screen. The oversize, when reduced to the proper fineness for flotation treatment, did not offer the least trouble and yielded a high-grade concentrate in the laboratory

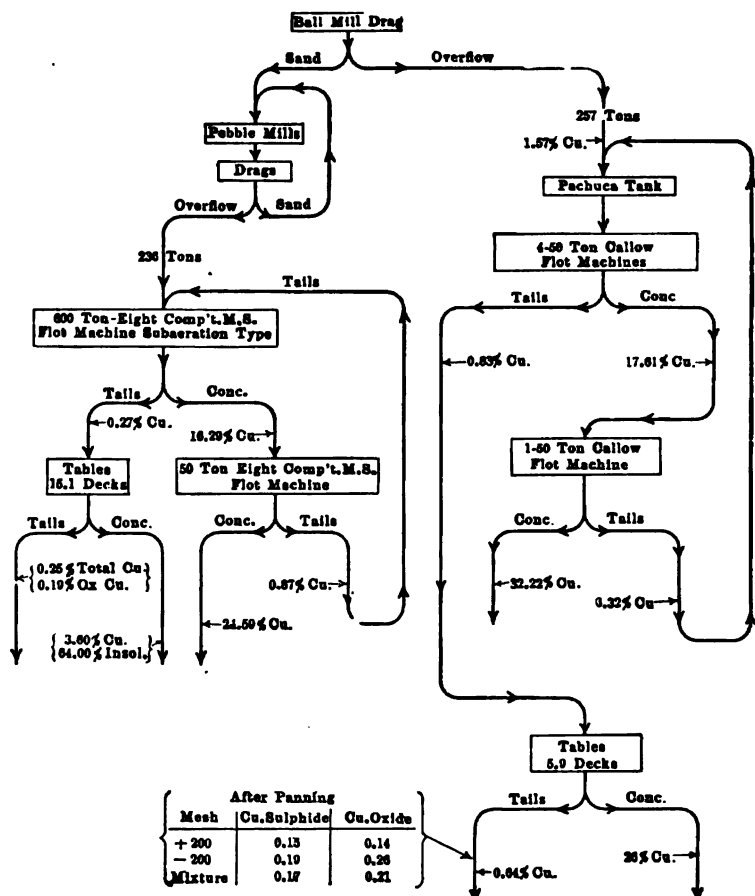


FIG. 68.—Tests to determine the effect of separate treatment of slime and reground sand.

machines, while the undersize proved extremely refractory (refer to Table 3). The same experiment was repeated on a large scale, the products from a Marcy ball mill crushing to about 6 mesh being treated on a drag classifier. The oversize, after being reduced in pebble mills to the necessary fineness for flotation treatment, was sent to one group of flotation machines, while the overflow from the drag classifier

TABLE 3.—SCREEN ANALYSIS OF ORE DRAWN FROM MILL BINS AND FLOTATION TESTS OF DIFFERENT SCREEN SIZES

Screen size	Weights, per cent.		Per cent. total copper	Copper contents	Per cent. oxidized copper	Con- tents ox. cu	Flotation results		Re-covery per cent.
	Cumul.	Indiv.					Tails	Midds. +conc.	
Inches									
+ 1	30.8	30.8	1.05	0.324	0.13	0.40	0.22	12.8	80.5
+ ½	47.9	17.1	1.34	0.329	0.11	0.19	0.22	17.4	84.7
Mesh									
+ 4	70.9	23.0	1.44	0.332	0.12	0.28	0.17	17.1	89.0
+ 8	75.2	4.3	1.42	0.061	0.12	0.05	0.17	16.3	88.8
+ 14	80.3	5.1	1.77	0.068	0.13	0.07	0.26	16.8	86.7
+ 28	84.4	4.1	2.14	0.080	0.14	0.06	0.61	16.3	74.3
+ 48	87.5	3.1	2.62	0.081	0.19	0.06	0.72	11.3	77.5
+100	89.9	2.4	3.29	0.079	0.24	0.06	0.63	22.5	83.1
+200	91.9	2.0	3.16	0.063	0.27	0.05	0.60	20.1	83.5
-200	100.0	8.1	1.53	0.124	0.43	0.35	1.57	1.48	None
		100.0	1.44	1.441	0.16	0.157			

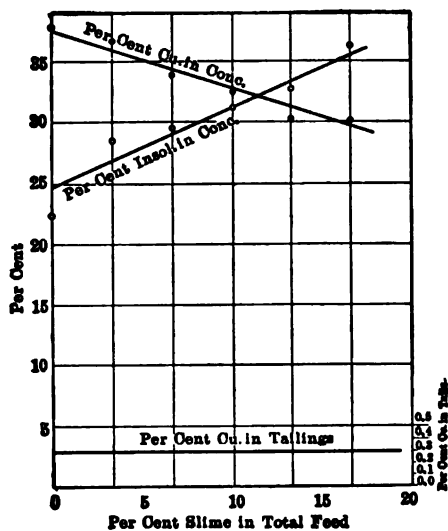


FIG. 69.—Influence on flotation of mixing primary slime and reground sand.

The + 1-in. and + ½-in. sizes resulting from the screen analysis were ground to 10 mesh, mixed with varying amounts of -200 material and the mixtures ground with the usual amounts of oil in the laboratory ball mills. After this preliminary treatment, the samples were treated in the laboratory flotation machines.

was sent to another group. Fig. 68 illustrates the results of this experiment. While the reground oversize yielded remarkable lean tailings, the product containing the primary slime could not be treated advantageously by flotation. We were almost on the point of concluding that in order to get the best results, a separation of the primary slime should be made in our large mill and flotation should not be entirely relied upon for the treatment of the slime. Before finally deciding on this point, however, some additional laboratory experiments were made to study the influence of the primary slime on flotation. These tests established one fact quite well, which is, that when refractory slime is mixed with a sufficient quantity of coarse ore ground to the necessary fineness, in other words, when the percentage of primary slime in the flotation feed is kept low, the slime loses its refractory character. I do not mean to say that by reducing the quantity of primary slime, the deleterious influence is reduced in proportion so that it cannot be detected as easily, but conclude from our tests that when the flotation feed contains a sufficient percentage of comminuted coarse rock, the primary slime contained in it can be treated more advantageously than it could by itself. Some tests bearing on this point are illustrated in Fig. 69. They indicate that as high an amount as 20% primary slime may be mixed with ground coarse ore without causing increased tailing losses. The concentrates, according to this set of tests, seem to have a tendency, however, to carry more insoluble matter with an increasing percentage of slime.

Influence of Iron on Flotation.—While these tests were in progress, we made another accidental discovery which proved very helpful to us. In our tests on the most economical way of reducing the ore to the fineness necessary for flotation we had, among other machines, a ball mill in competition with pebble mills. In the ball mill, steel balls performed the duty that in pebble mills was done by flint pebbles.

For a while, the ball-mill discharge was treated on one flotation machine, while the pebble-mill discharge was treated on a group of others. While this flow sheet was being followed, we thought we noted that a flotation machine treating the ball-mill product, showed the influence of the primary slime to a lesser extent than the flotation machine treating the pebble-mill product. In a discussion with Dr. Ricketts and Mr. Mills, the question was raised as to whether the iron introduced in the mill pulp by the attrition of the balls might not have something to do with the fact. The question was accordingly made the subject of some laboratory experiments. The results of a series of such experiments are represented in Table 4, and proved conclusively that the iron had a beneficial influence on flotation in counteracting the harmful effect of the primary slime. This discovery was one of the inducements for installing ball mills in the big concentrator, while originally pebble mills had been considered for this purpose.

We have not yet reached a point where we can safely give the reason

for the action of the iron introduced into the flotation pulp. It is sure, from the experiments referred to, that the same results, as by grinding with balls, could be obtained by introducing the iron in finely divided form, say in the form of filings, into a pebble-mill pulp. We supposed for a while, and I am not yet certain that this supposition is incorrect, that the metallic iron might react on the impurities contained in solution in the mill water and introduced therein with the primary slime. We find, as a matter of fact, that our ores contain very little in the nature of

TABLE 4.—EFFECTS OF IRON AND OTHER SOLIDS ON THE FLOTATION OF REFRACTORY COPPER ORES

Test No.	Grams ore	Per cent. copper	Grams copper	Concentrates			Recovery, per cent.	Remarks
				Grams	Per cent. Cu	Grams Cu		
F21	750	2.01	15.07	45	23.66	10.65	70.7	Added 10 g. iron filings.
F22	750	2.01	15.07	47	29.52	11.52	76.4	Added 10 g. iron filings.
F27	750	2.01	15.07	43	27.10	11.65	77.5	Added 2 g. iron filings.
F28	750	2.01	15.07	47	26.90	12.64	84.0	Added 2 g. iron filings.
F45	750	2.01	15.07	31	23.84	12.16	80.7	Added 10 g. iron filings.
F46	750	2.01	15.07	48	25.60	12.29	81.5	Added 10 g. iron filings.
F47	750	2.01	15.07	27	8.80	2.38	15.8	Blank with no solids added.
F48	750	2.01	15.07	28	8.34	2.34	15.5	Blank with no solids added.
F49	750	2.01	15.07	62	20.14	12.49	83.0	Added 10 g. miscellaneous iron filings from shops.
F50	750	2.01	15.07	63	20.10	12.66	84.0	Added 10 g. miscellaneous iron filings from shops.
F51	750	2.01	15.07	60	18.82	11.29	75.0	Same as F49 and F50 by different observer.
F52	750	2.01	15.07	62	19.54	12.11	80.5	Same as F49 and F50 by different observer.
F53	750	2.01	15.07	29	6.16	1.79	11.9	Blank with no solids added.
F54	750	2.01	15.07	30	7.96	2.39	15.9	Blank with no solids added.
F55	750	2.01	15.07	59	21.08	12.44	82.7	Added 10 g. iron filings.
F56	750	2.01	15.07	56	25.80	14.45	96.0	Added 10 g. iron filings.
F64	750	2.01	15.07	65	19.52	12.69	81.9	Ground in mill with steel balls instead of pebbles.
L27	750	1.10	12.75	37	28.42	10.51	81.7	Blank on good flotation ore.
L28	750	1.10	12.75	34	28.46	9.85	78.3	Blank on good flotation ore.
L29	750	1.10	12.75	76	5.66	4.30	33.4	Identical conditions as L27 and L28 but added 10 g. zinc filings.
L30	750	1.10	12.75	83	5.06	4.20	32.9	Identical conditions as L27 and L28 but added 10 g. zinc filings.

soluble salts, and that whatever they do contain is mainly confined to the primary slime. For this reason, in laboratory tests we have tried repeatedly to substitute pure water for the water contained in the mill pulp. In every case, we have noted some improvements in the results obtained. We have also found that when we separate the water from the refractory pulp, treat it with iron filings, and add it to the original

pulp again, we get a certain improvement in the recovery, but we have not been able to get an improvement equally as good as that obtained by direct introduction of finely divided iron into the pulp. For this reason, we have often thought that the effect of iron is physical rather than chemical in character. The iron exists in the pulp, at least partly, in the metallic form, as can be proven by the use of the magnet. If necessary, the effect of the iron could be increased by removing the iron contained in the tailings pulp by means of electromagnets and returning it to the mills or the flotation machines. Although we have considered the possibility of applying this fact commercially, we have not yet done so.

I have been told by several people that they have tested the influence of finely divided iron on their ores and have not obtained any improvement whatever. This shows, evidently, that metallic iron is not a universal remedy for all flotation troubles, but as far as our primary slime is concerned, our experience leaves no doubt about its usefulness, and I believe the figures quoted above are positive enough to bear out my statement. As a curiosity I have included in Table 4 some figures showing the influence of the addition of zinc filings to ore that without the addition does not offer any difficulty in the flotation treatment.

Influence of Chemicals on Primary Slime.—We also found that the harmful influence of primary slime can be counteracted by the introduction of certain chemicals; for instance, we have used advantageously sodium hydroxide or potassium cyanide. The use of sodium hydroxide was recommended to us by the Minerals Separation Co., which, I understand, applied it first in the plant of the Caucasus Copper Co.

Tests to determine the most advantageous fineness for treatment of the ore according to our flow sheet, *i.e.*, flotation machines followed by tables, naturally formed an essential part of our tests. Results of this character seem to indicate that to get the best results grinding should be carried to such a point that not more than about 1 or 2% of the pulp will remain on a 48-mesh screen (Tyler type). The grinding at our large concentrator is at present carried closely to this point.

Operation of Large Concentrator.—The first four sections of the Commercial Concentrator of the Inspiration company were equipped with Callow flotation machines. As will be seen from this illustration, the ore, after having been reduced to the desired fineness, and having been oiled in the Marcy ball mills, is sent to 12 (in other sections 8) Callow cells. The tailings from these primary cells are sent to a drag classifier designed by Mr. Burch, which effects a separation of sand and slime. The slime is retreated in 12 (in other sections 16) additional Callow cells, while the sand is sent to a hydraulic classifier and then to tables. The concentrates made both on the primary and the secondary Callow cells are retreated in a group of four Callow cleaning cells from which

the tailings are returned to the head of the primary Callow cells. On the tables of the Deister Machine Co. a separation is made between concentrates and tailings, but no middlings are produced and no part of them is reground. This, however, is probably only a temporary arrangement, the idea being that a suitable regrinding and reconcentra-

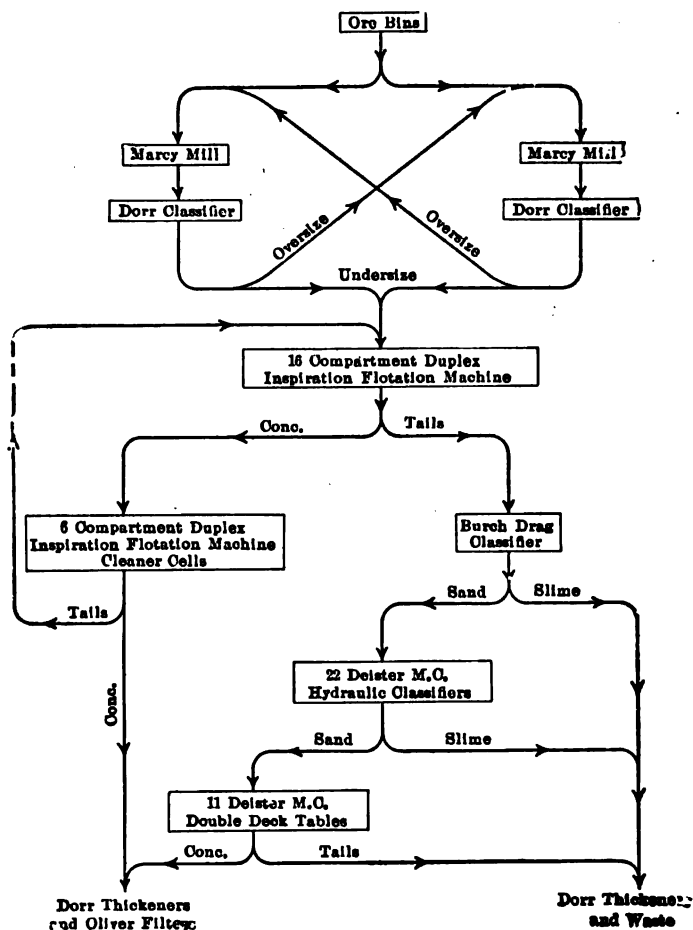


FIG. 70.—Flow sheet of Inspiration concentrator sections equipped with Inspiration flotation machines.

tion process will be installed later, after having been worked out by extensive tests.

Flow Sheet of Inspiration Sections.—Thirteen sections of the Inspiration concentrator are equipped with Inspiration flotation machines. The flow sheet of these sections is represented in Fig. 70. The ore, after

having been crushed and oiled, passes to the head of the flotation machines and traverses their whole length. The resulting tailings are split, by the same kind of a drag as mentioned before, into a sand and a slime product. The slime product is run to waste. This is thought permissible because, after having passed through the whole length of the flotation machine, the slime has been impoverished to such an extent that retreatment seems unnecessary. The sand product undergoes the same retreatment as that described for Callow sections.

Inspiration Flotation Machine of Steel Construction.—Fig. 71 shows the drawing of an Inspiration flotation machine of steel construction as designed by Mr. Burch's office. At the end of each section of eight compartments, a pulp overflow is provided which regulates the level of the pulp in the preceding compartments of the flotation machine. The subdivision into two sections of eight is made in the interest of closer regulation of the pulp levels. It had been feared that if no such subdivisions were made, and the regulation of the levels accomplished at the tail end only, appreciable fluctuations might take place in the compartments near the feed end due to changes in the volume of pulp treated, and might prove highly undesirable.

One point in the construction of the Inspiration flotation machines in which we take considerable pride, is the method in which the air is introduced. The machines have solid bottoms; all pipe connections are made from above and are at all times visible. The porous bottom rests on top of the regular bottom of the machine and is introduced from above. It is illustrated in Fig. 72. As will be seen, it consists of a lower air chamber having partitions running lengthwise, which, however, do not extend down to the bottom of the air chamber and, for this reason, do not interfere with the interchange of air between the narrow compartments thus formed. A grate, which forms the upper part of the porous bottom, can be fastened to the lower part by a number of bolts. Before putting them together, a porous medium (for instance, canvas) is placed between the two pieces. The air enters through a pipe from the top which screws into the lower casting. Arrangements are made, of course, to secure an air-tight joint where the pipe passes through the porous medium. When assembled, the bars of the upper grate form channels in the lengthwise direction of the machine. The bars are the only parts that protrude above the canvas and, on account of their arrangement longitudinally, they do not interfere with the passage of pulp through the machine.

The cleaner cells consist of six compartments in series. Both rougher and cleaner cells are divided by a partition in the middle, running the whole length of the machine. The operator is therefore, enabled to throw all the feed on one side of the machine when desired, thus permitting repairs to be made on the compartments of the other side whenever necessary.

In the beginning, it had been supposed that throwing all the feed on one side of the flotation machine might seriously overload that side, and for this reason, in the first sections installed, provisions were made to permit removing and replacing the porous bottom from any one compartment without interrupting the flow of pulp. As far as lifting out the

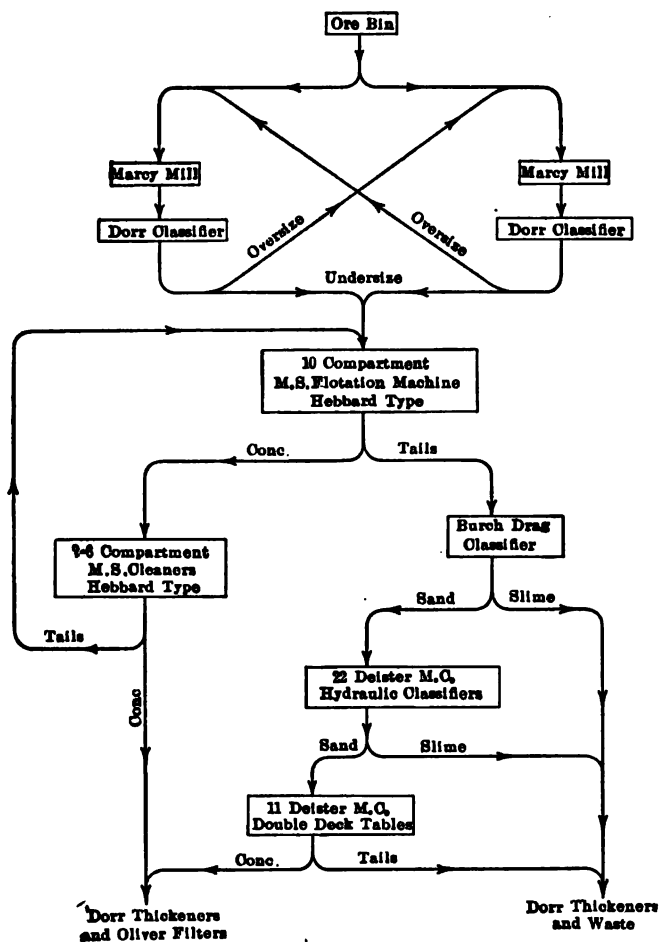


Fig. 73.—Flow sheet of Inspiration concentrator section equipped with Minerals Separation machines.

bottom is concerned, this, of course, can be done in any case, but while one of the bottoms is withdrawn sand will pile up to a certain extent in the middle of the compartment. To remove this sand before returning the porous bottom, a system of perforated water pipes was installed

under the air chamber. They allow the washing out of the bottom of the machine by the application of water pressure.

One section of the concentrator is equipped with machines of the Minerals Separation Co., Hebbard type. The flow sheet, as illustrated in Fig. 73, is practically the same as the one shown in the Inspiration sections. The only difference is, that two cleaner cells in parallel are used in place of the one cleaner cell of the Inspiration type.

Filter Plant.—It is necessary to reduce the ore to a certain fineness before the mineral contents of the feed can be treated by flotation, and as fine concentrates have the characteristic of retaining water with greater tenacity than coarse concentrates, a filter plant is a necessary adjunct of a flotation plant. In the Inspiration mill, both the flotation and table concentrates are sent to Dorr thickening tanks, five of them being 60 ft. in diameter and three of them 80 ft., representing a total

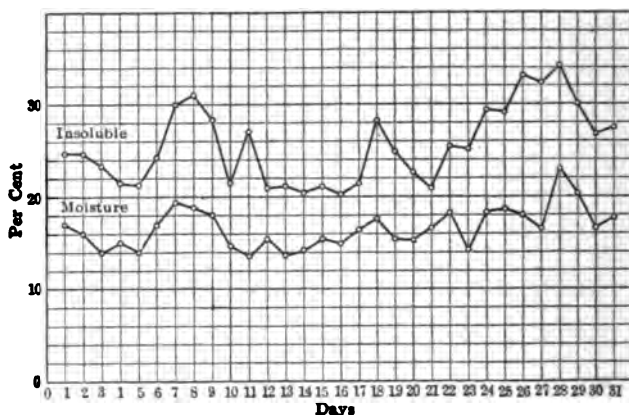


FIG. 74.—Curves showing relation between moisture contents and insoluble matter in concentrators. October, 1915.

area of about 29,217 sq. ft., or, as the daily production of concentrates amounts to about 600 tons, a settling area of around 48.7 sq. ft. per ton of concentrate sent to the settling tanks. The settling tanks are each provided with a double ring of high baffle boards to prevent the foam that forms on the top from contaminating the overflow. The settling used to be carried out in two steps, some of the tanks serving as preliminary settling tanks, while others resettled the overflow from the preliminary tanks. At present we operate all the concentrate settling tanks in parallel. The spigot discharges feed six Oliver filters. The ratio of solids to water about 1.65 to 1 at this point. The filters reduce the concentrate moisture to an average of about 17% of the wet pulp. This figure varies, however, within rather wide limits from day to day. The moisture contents of the concentrate seem to depend in the first place on

the percentage of insoluble material contained in the concentrates, as will be seen from the graphic representation of the relation between the two things in Fig. 74. It will be noted that a rise in the percentage of insoluble matter generally corresponds to a rise in the concentrate moisture and *vice versa*. A moisture of 17 % is, of course, in excess of what seems desirable in view of the fact that the smelter penalizes water contained in the concentrates. For this reason, several attempts have been made to reduce the moisture. The first plan followed was to inject steam into the filter tank, with the object of heating the pulp. An appreciable increase in the capacity resulted from this, while no marked reduction of the moisture contents took place. The same effect was obtained by the addition of slacked lime, which we tried lately in our Oliver filters. It shows itself very readily in the formation of a thicker filter cake, thus causing a larger output of the filters. We have, however, not been able to effect a reduction in moisture by the use of lime. We have adopted the addition of lime to the filter pulp in regular operation on account of its decided benefit as far as the capacity is concerned. As a rule, five filters can handle the whole output of the concentrator; that is, each 12 by 12-ft. Oliver filter has a capacity of about 120 tons in 24 hr.

Results from the Commercial Concentrator.—On account of the fact that the concentrator operated by the Inspiration company has only just left the stage of construction, it is not possible to give results obtained in the operation of it as a whole. As, however, an interval of several months has elapsed between the starting of the first and the 18th section, we are in a position to give at least preliminary results derived in actual operation on a fairly large scale.

Each of the sections of the Inspiration concentrator has a rated capacity of 800 tons; at present most of the sections exceed 900 tons actual capacity. Ordinarily, this tonnage is treated on one Standard duplex Inspiration machine consisting of 16 double compartments, while the concentrates are cleaned on a cleaner cell with six double compartments. The size of the roughing compartments is 3 ft. by 4 ft. 3 in.; the size of the cleaner compartments is 3 ft. by 3 ft. Accordingly, the total area of combined rougher and cleaner cells is 492 sq. ft. Assuming the tonnage handled to be 800 tons, this makes a tonnage treated of 1.62 tons per day per sq. ft. of surface. In the course of construction, the erection of flotation machines did not always keep pace with the erection of the grinding mills. For this and other reasons, it has frequently been necessary to combine the product from the two grinding sections in one flotation section; in other words, to throw an overload of 100 % on the flotation machines. Our experience is, that overloading the machines by 100 %, or raising the tonnage to 3.24 tons per sq. ft. of surface, does not increase the tailing losses very materially.

It follows, therefore, that the machines handle an overload without serious additional copper loss, and it has been our experience that there

is no mechanical trouble whatever, connected with overloading the flotation machines to such an extent. We have also had to resort to overloading the Callow cells, and have likewise found that the recovery does not fall off seriously.

Air and Power Consumption.—We have found that three blowers, of a capacity of 8,000 cu. ft. per minute each, are sufficient to supply air for six sections of flotation machines. Accordingly, one section requires 4,000 cu. ft. of air per minute. As the flotation machines in an Inspiration Section have a surface of 492 ft., it follows that the air consumption per square foot of porous surface is 11.8 cu. ft. per minute; the maximum pressure required is $4\frac{1}{2}$ lb. at the blowers and practically the same at the flotation machines, as there is no serious loss of pressure in the air conduits. The power consumed for furnishing air has been determined carefully by wattmeter readings of the motors furnishing power for the blowers. The actual power consumed is 87.5 kw. per section; or, figuring again with the rated section capacity of 800 tons, 2.63 kw.-hr. per ton of ore treated. We have not been able to make separate tests for the Callow and Inspiration sections, but assume that the air consumption is approximately the same in both. We have no figures yet for the Hebbard type machine. To get the total amount of power consumed in connection with the flotation machines, a small amount of power, probably around 10 kw. or 0.30 kw.-hr. per ton of ore, has to be added to the power consumed for the production of air on account of the fact that the cleaner tailings are pumped back to the rougher cells and retreated, and that in the Callow sections the slime overflow of the drag classifiers is repumped to the secondary flotation cells. In the Inspiration sections, 0.425 sq. ft. of actual open porous surface and 0.615 sq. ft. of flotation machine area are provided for the treatment of 1 ton of ore in 24 hr.

The total floor space of the flotation floors amounts to 25,200 sq. ft., or 1.75 sq. ft. per ton of ore treated in 24 hr. (rated capacity). The total floor space occupied by the concentrator proper (not including, however, the crushing plant located at the mine) is 81,900 sq. ft. or 5.68 sq. ft. per ton of ore, and the total floor space, if the settling-tank installation outside the main mill building is also included, is 250,890 sq. ft., or 17.4 sq. ft. per ton of ore (rated capacity).

The water added to the pulp where it reaches the flotation machines is about 3 tons of water for each ton of solid pulp. The subsequent treatment, in our case, requires a further addition of nearly 3 tons of water per ton of dry ore. However, as it is carried out on pulp that has been deprived of its slime contents by passing it over drag classifiers, the water thus entering the table tailings can be easily removed again. This is accomplished in our case by settling the table tailings separately from the slime tailings. The settling of the sand tailings is effected in a settling tank of home construction, consisting of a rectangular box with a sloping bottom into which a number of Caldecott cones are inserted.

The resulting overflow is not quite clear, but is made so by resettling in three 60-ft. Dorr tanks.

Out of the total quantity of water actually added to the ore during its passage through the mill, amounting approximately to 6 tons of water per ton of ore treated, as mentioned above, nearly 3 tons, *i.e.*, the amount added during the table treatment, can, as explained above, be reclaimed rather easily. For the reclamation of the rest of the water, settling ponds, of the well-known construction in which the reservoir is formed by building a retaining wall of sand across a gulch, are resorted to in addition to Dorr thickening tanks.

Recovery.—The recovery obtainable by the application of our milling process is determined entirely by the composition of the ore, *i.e.*, by the relation between sulphide and oxide copper contained in the ore. Our average sulphide copper extraction has been 90.39% for the months of March, April and May, 1916, the last months for which figures were available at the writing of this paper. A certain recovery of the oxide-copper minerals, especially carbonates, is made in the flotation process as well as in the gravity concentration process. The percentage of such mineral recovery is low, probably around 25%. For this reason, the recovery obtainable on ore containing a high amount of oxide, such as surface ore, is correspondingly lower. We have worked in the laboratory with the object in view of increasing the oxide recovery; for instance, by adding certain chemicals to the flotation pulp. An account of the results obtained will be found below, but we have not yet applied this method to an operating scale, nor have we decided on using one of the other methods applicable for this purpose, such as leaching.

Table 5 gives an average screen analyses of the feed and the general tailings of the Inspiration concentrator for the months of March, April and May. A segregation is made in the copper assay between sulphide and oxide copper, because, considering the present stage of the art, we feel satisfied with our mill work whenever the sulphide copper content of the mill tailings is low. As will be seen from the tabulations, a better recovery is made on the -200 material than on the coarser constituents of the ore, which proves the point that for ores of the character of Inspiration ore sliming is not to be feared since the introduction of the flotation process.

Oil Consumption.—Experience has shown that in the flotation treatment of our ores, we consume flotation agents up to 1½ lb. per ton of ore. At present, the oil mixture contains around 95% crude coal tar and a little less than 5% of oils derived from the dry distillation of wood.

The different tars that we have tested during the operation of our mill have shown greatly varying qualities as far as their flotation value is concerned. The first tar that we tested and made use of was home-made from domestic coal, and happened to be a very serviceable flotation agent. Since that time, we have tested tars from several States

and have found some that are suitable for our purposes, while others are less so, and still others even entirely unsatisfactory. We have obtained satisfactory tar products from New Mexico, Colorado, Missouri and Illinois. These States furnish at present as much as we need for our consumption. For awhile, it seemed possible that we might have to import from a long distance the large quantities of tar that we require. During that period we tried to find substitutes, and looked especially toward the utilization of fuel oil for this purpose, but we have not been able to get as good results with any kind of fuel oil as with crude coal tar.

TABLE 5.—AVERAGE SCREEN ANALYSES FOR THE MONTHS OF MARCH, APRIL AND MAY, 1916, OF FLOTATION FEED AND GENERAL TAILS

Flotation Feed					General Tails							
Mesh	Per cent. Weight		Copper contents		Per cent. weight		Sulphide copper contents		Oxide copper contents		Total copper contents	
	Cum.	Indiv.	Per cent.	Grams	Cum.	Indiv.	Per cent.	Grams	Per cent.	Grams	Per cent.	Grams
+65	9.5	9.5	0.45	0.042	9.5	9.5	0.18	0.017	0.12	0.011	0.30	0.028
+100	21.2	11.7	0.86	0.101	21.2	11.7	0.19	0.023	0.14	0.016	0.33	0.039
+150	33.5	12.3	1.91	0.235	33.5	12.3	0.11	0.014	0.19	0.023	0.30	0.037
+200	39.2	5.7	2.69	0.154	39.2	5.7	0.14	0.008	0.19	0.011	0.34	0.190
-200		60.8	1.85	1.125		60.8	0.06	0.036	0.47	0.286	0.53	0.322
Totals		100.0		1.657		100.0		0.098		0.347		0.445
Assay direct			1.62				0.101		0.318		0.419	
Oxide.			0.36									

Our experience is, that we can get along with coal tar alone. It is beneficial, however, to add wood-distillation products in small quantities, for instance, those containing pine oil. While coal tar makes a very strong and heavy froth, such as appears to be required to keep coarse mineral particles in suspension, the wood-distillation products have the characteristic of producing a multitude of froth bubbles, such as seem necessary to furnish the large surface required to save the very fine mineral particles. Because the finer ore particles, as (slime) on account of their small diameter, expose a large surface, it is evidently necessary to produce a correspondingly large surface of froth in order to save them by flotation.

Operating Cost for Flotation.—The number of men necessary for the operation of large flotation machines is remarkably small. At the Inspiration plant, one operator supervises four sections of flotation machines. Two Mexican helpers assist him in washing the bottoms, thus insuring a free passage of air through the porous medium. At the prevailing high prices of American and Mexican labor, this means an expense of somewhat more than 1.5c. per ton of ore treated. The total expenses representing flotation proper were as follows for the months of March, April and May, 1916:

	Cents per ton
Labor.....	1.62
Flotation oils.....	1.65
Other supplies.....	0.35
Power.....	2.14
Total.....	5.76

The subsequent table treatment of flotation tailings, the filter treatment of the concentrates and other operations connected with the process of concentration, belong more or less to flotation treatment, and their expense should also be considered when the cost of the flotation process is to be established. The total milling cost, exclusive of crushing and grinding, has been for the past few months in the neighborhood of 20c. When the cost of crushing and grinding is included, the cost is about 40c. per ton of ore. Royalties for the use of the flotation process are not included in any of these cost figures.

Discussion of Results Obtained.—The criticism has occasionally been raised against the Inspiration company that they were slow in deciding on the design of a concentrator. The reason for such slowness was, of course, that the development of the flotation process coincided with the period during which the fundamental points in the design of the concentrator had to be settled. The management, therefore, thought it best to be very conservative in installing standard concentrator equipment, which, in case the new process held what it seemed to promise, might prove to be entirely superfluous. The main point of interest, therefore, is whether it was wise for the company to wait instead of installing a gravity concentrating plant as had been originally considered.

As far as the comparison of a flotation plant, as at present operated by the company, with the gravity concentration plant, as originally considered, is concerned, the recoveries actually obtained in the company's flotation plant are so much higher than those indicated by tests in the original test mill as being obtainable in a gravity concentration plant on ore of the same character, that there cannot be the least doubt which system is the better.

In discussing the second question which comes up in this connection,

whether, for ores of the character of Inspiration ore, gravity concentration supplemented by flotation would be preferable to the simple flotation process that has been adopted by the Inspiration company, the crucial point is this: Can the slime be reduced to lower copper contents by a combined process than by the process followed here? The reduction of the copper in the sand is, as mill men know, simply a question of fine grinding combined with suitable concentration.

In support of the principle followed by the Inspiration company, I desire to call attention to the screen analyses of our general tailings represented in Table 5. They show that the sulphide copper left in the -200-mesh material is very low.

In addition, the floor space required for a plant like this is materially smaller than for a plant of the combination plan, which also means that the construction cost is much lower; and further the operating costs are low by reason of the greater simplicity of design. I hope these facts are sufficient to decide the point at issue as far as the ore of this district is concerned.

I hope that our neighbors of the Miami company, whose mill was built before the flotation process was known, will uphold me in my statements. If this point is admitted, it will endorse the principle followed in the design of the Inspiration mill, of doing away with reduction in stages and concentration in stages.

It must be granted that the settling of flotation-concentrate pulp requires extensive floor space, as shown by the figures referred to above, and that settling and subsequent filtering absorb a certain fraction of the mill operating costs. However, as a ratio of concentration in the Inspiration mill is 25 into 1, only $\frac{1}{25}$ of the ore is thus treated. The actual expenses increase, of course, proportionately for ore which does not permit concentration with such a high ratio as the ore existing in the Inspiration mine. Therefore, there will be a point beyond which a different system of concentration is preferable. Just what the critical ratio of concentration is, must be determined by calculation and testing based on individual conditions.

In summing up I want to say that the fact that the Inspiration company has been able to design a commercially successful flotation plant and has found ways that hold out prospects of raising the plant to a very high state of efficiency, must be attributed to the policy followed by the company of spending great sums of money for the purpose of investigating the flotation process on a commercial scale. In carrying out these investigations, a close coöperation between laboratory and operating force helped us, I believe, more than anything else. I would like to give credit to each person who had a share in contributing toward the success of the work, but cannot do it, because the list would be too long.

CHAPTER XVIII

THE PLACE OF FLOTATION IN METALLURGY

In view of the application of the flotation process to so many different kinds of metallurgy, its usefulness has apparently been shown to be exceedingly great. It is, up to this time, being applied to ores containing copper, zinc, lead, silver, molybdenum, and some other metals, in addition to the ores of gold and silver alone. In spite of this wide application, however, its definite position has not yet been taken. The ideas of metallurgists are continually on the change, and it is impossible at this time to forecast what the final field of the process will be. An authority who has been connected with the process since its first introduction into this country, and who has studied it carefully in many fields, predicts that its application will be just as wide as concentration in any other form. There is much reason to believe that this estimate is probably right.

Development at Anaconda.—In 1914 the technical press noted a plan for metallurgical improvement that had been inaugurated at Anaconda. At that time the contemplation of the Anaconda metallurgists was to go into copper leaching on a large scale, but their attention had just been drawn to the possible economic superiority of the flotation process, which might be substituted for mechanical concentration, and it was suggested that they stood ready to abandon the leaching feature of their programme in part or in whole, if that should appear desirable. Developments have shown that it was desirable, and it turned out that only the leaching unit then under construction was completed, it fitting well for a special purpose, while the main Washoe concentrator has been changed into a flotation plant, whose operation has already been noted in this volume.

The operation of this plant was commented upon editorially in the *Engineering and Mining Journal* of Nov. 13, 1915, which calls attention to the fact that in September, 1915, about 8,000 tons of ore daily were under treatment by the new process. By applying the data derived from that month to the production

figures of 1913, and computing on the basis of 14c. per lb. for copper, it was estimated that the saving would have been about \$7,250,000. Putting the thing in a different way, the gain of valuable metals previously lost in tailing and slag would have been 50,000,000 lb. of copper, nearly 1,000,000 oz. of silver, and about 5,000 oz. of gold. This enormous saving is being effected at the cost of about \$5,000,000 for the new plant. In other words, it is expected that the vast sum of money will be repaid in about eight months.

This saving has not been altogether due to flotation, but it is largely there. Formerly the tailing wasted at the mill assayed about 0.62% copper, while now it is only about 0.1%, and there is reasonable expectation that even that small figure will be reduced. The Anaconda ore fed into the mill contains 60 lb. of copper per ton, of which only $1\frac{3}{4}$ lb. now goes away in the tailing. The recovery is therefore about 97 $\frac{3}{4}$ %. Other great items of the saving are the improvement of the reverberatory smelting process by the introduction of coal-dust firing; the abandonment of briquetting and blast-furnace smelting in favor of the new reverberatory process, and the saving in freight resulting from the direction of all the concentrating ore to Anaconda, instead of sending some of it to Great Falls. Some saving and profit has been through the operation of the leaching plant.

Another expression of opinion upon the subject is given by W. R. Ingalls in an article on "Electrolytic Zinc," appearing in the *Engineering and Mining Journal* of March 4, 1916. Mr. Ingalls says:

"Another new and uncertain factor is the bearing of flotation process of ore concentration upon the metallurgy of zinc. I think this had a good deal to do with the institution of the Anaconda work. About all metallurgical work is a sequence of steps of concentration and refining, treating the bulk of the ore by a cheap but wasteful process, and delivering the concentrated product to a more costly but less wasteful process. Now, most experimenters in the hydrometallurgy and electrometallurgy of zinc heretofore have contemplated the application of a costly process to run of mine ore. The flotation process has enabled ore to be concentrated at relatively small cost with but slight loss. Let it be observed, therefore, that Mr. Laist is applying his costly process not to run-of-mine ore, but to flotation concentrate in which about 90% of the zinc is contained in about one-fourth of

the original weight. This is the new and important feature of recent zinc electrolytic work.

We must let our thoughts run a little farther ahead. The treatment of flotation concentrate is one of the present troubles of the zinc smelter, owing to its excessive fineness, which produces difficulties that it would take too long to describe. Yet the proportion of this class of ore that the zinc smelter is getting is still relatively small. The supply of it is, however, bound to increase, and when it becomes large the troubles of the zinc smelter will really begin."

The mechanical disposal of flotation concentrates has been discussed by R. J. Anderson.¹

The Handling of Concentrates.—Flotation concentrate, *i.e.*, the froth entangling the valuable sulphides (and gangue in subordinate amount) plus water, is an unsmeltable product as such. The first step in the disposal operations requires a breaking of the froth, particularly if it be a mechanical froth, such as is produced by the Minerals Separation, Hebbard-Harvey, Janney, or similar mechanically agitated machines. The pneumatic froth of Callow or Flynn-Towne is ephemeral and breaks of itself when removed from the influence of the injected air (although, of course, its stability is influenced by the kind of oil used), and is, therefore, more easily handled. The mechanical froth, however, is thick and more or less permanent, and its disposal has presented serious difficulties.

A common practice for froth breaking, which is quite effective, simply calls for the spraying of water on the froth as it flows in its launder. In a number of instances bucket elevators have proved quite efficacious; the splashing and striking of the buckets causes the froth to break up. At the Herculaneum plant of the St. Joseph Lead Co. there have been installed pug-mills for this purpose. In addition, these mills insure a steady stream of concentrate feed to the mixing belt whereby the flotation product is mixed uniformly with the usual feed for the Dwight and Lloyd sintering machines. The method of running the concentrate over tables of the Wilfley or Card type has been tried and is successful in breaking up the froth. Froth may also be effectively broken by the addition of reagents such as acid or lime, or even by adding more oil. In regard to this latter, if an excess of oil is added to a successfully operating flota-

¹ *Met. and Chem. Eng.*, April 1, 1916.

tion unit the froth may be practically "killed." A filter of the pressure type will break up the froth in the dewatering operation.

Dewatering the Concentrate.—All flotation processes produce a wet, sloppy, frothy concentrate which is extremely difficult to handle and ship unless dewatered. The skillful and economical disposal of flotation concentrate preparatory to shipment and smelting is, therefore, an important problem. In discussing dewatering practice, certain machines have found their way into the mills and certain methods have been adopted which probably embody the tendencies toward a future standard practice. Some of the apparatus used for dewatering includes settling bins, filters—pressure or vacuum, drag classifiers, screw classifiers and thickeners.

The practice of settling the froth in bins, while a common one, does not reduce the moisture sufficiently low for shipment by rail unless for short distances. The mill of the Utah Leasing Co. at Newhouse, Utah, has large bins of 200-ton capacity for settling the concentrates. Practically all of the larger plants have come to the use of filters either of the pressure or vacuum type, or both.

The functional utility of the filter lies in its ability to reduce the moisture content of the material treated to practically any desired percentage. Pressure filters of the Kelly type can take the flotation froth direct and perform the function of froth breaking and dewatering in the same operation. The pressure filter is more bothersome to operate, because it is intermittent in action and requires constant attendance. Although the pressure filter can reduce the moisture content to as low as 6%, the vacuum type finds more favor. This type of filter is not well adapted to the immediate dewatering of froth concentrates because such product will not cake well on account of its high percentage of moisture—60 to 70% and occasionally considerably more. The pulp feed for vacuum filters should have a ratio of 1 to 1. If the froth be first broken and then dewatered in a Dorr tank, the vacuum filter can be used, and it is preferable to the pressure type by reason of its continuous operation and low operating cost. The vacuum filter in practice can make concentrates of any moisture content to as low as 8 to 10%. Where concentrates are to be briquetted, such a low percentage of moisture is not necessary, but should be as high as 15% or more.

An example cited recently in the current technical literature, which gives a method of dewatering flotation concentrates, is the practice at the Inspiration mill in Arizona. Here the concentrates from the flotation processes and from the tables converge to a drag-classifier; the coarse product from this machine is sent to an Oliver filter and the fine product to a V-shaped settling tank, where it is thickened. The thickened and settled fine from the tank then goes to the filter. The filter cake from the Oliver machine is trammed to a bin and thence loaded into bottom-discharge railroad cars for shipment to the smelter. At the Miami mill in Arizona, the flotation concentrate is thickened in Dorr tanks and the thickened product then conveyed by bucket elevators to a group of 12-ft. $\times$ 12-ft. Oliver filters. The caked concentrate is scraped off over a steam-heated plate and then removed to a steel storage tank preparatory to shipment to the smelter.

A type of machine brought out in recent years by one of the Western companies which is applicable to the dewatering flotation concentrates is the so-called Ovoca classifier. This is a classifier of the screw type. The froth is broken down effectively, giving a continuous clear overflow solution at the weir and a continuous discharge of dewatered concentrate—moisture 15 to 25%. A screw dewaterer is in operation at the Oneida Stag mill at Idaho Springs, Colo. At this place the flotation concentrate plus the table concentrate is passed through a screw dewaterer and the two are thus mixed and dewatered in one joint operation. The dewatering machine removes from 60 to 70% of the slimes with the coarser table product, and the overflow removes the balance of the slime. This latter goes to settling boxes and is there settled and thickened.

The practice of washing flotation concentrate is a development as yet new in American practice. The leading consideration in washing is the removal of part of the insoluble matter whose presence in the final concentrate detracts from its grade. Washing is employed at a New South Wales mill in Australia. Here the flotation product is led into settling bins and washed by means of spray water playing on it as it enters. The effect of this is to disintegrate the froth so that the sulphides sink to the bottom and at the same time part of the floated gangue is carried over the lip by the moving stream of water.

Drying the Concentrate.—Drying is an important consideration, particularly when the concentrate has to be shipped long distances. Every pound of water in the shipped product has to be paid for as freight, hence should be as low as possible. The advantages of dewatering and drying lie in reducing the water content, and therefore the freight charges; and preventing loss of slimy concentrate due to leaking tanks and cars. It also avoids the trouble due to freezing in cold weather.

At Engels, Calif., the flotation concentrate is settled and thickened outside the mill. The thickened concentrate is then filtered in a machine of the Oliver type, making a cake containing 10 to 12% moisture. To reduce the moisture to 5 to 6%, because of the long shipping distance, an unused drag classifier has been remodeled so that heat can be applied beneath it, and the material dried on it. The dried concentrate is sacked, hauled to the railroad, and the sacks then emptied into railroad cars for shipment to the smelting plant.

Flotation and Cyanidation.—In discussing the comparative values and fields for flotation and cyanidation, one of the important questions that arise is how the flotation concentrate, when made, is to be treated, particularly in isolated plants. It may be desirable in many cases to treat this material on the ground where it is produced, and cyanidation has been proposed, and sometimes used, as a process for doing this in the treatment of precious-metal ores.

It is evident that trouble may possibly occur through the oil contained in flotation concentrates, that may be detrimental to cyanide treatment, possibly acting as a cyanicide. Some operators have made actual tests, and in fact are treating flotation concentrates from precious-metal ores without any difficulty, and one operator has announced that no difficulty was found in such a system of operation. In other cases, however, great difficulty has been found, the consumption of cyanide proving altogether abnormal. The difficulty then, apparently does exist sometimes, but not always. In connection with this subject, Professor Clevenger, of Leland Stanford University,¹ points out:

“The remedies suggested have been saponification of the oil film by preliminary treatment with a caustic alkali; washing with a light mineral oil, such as gasoline, which is presumed to

¹ *Met. and Chem. Eng.*, June 15, 1916.

act as a solvent for the oil film; and various other expedients which have been tried more or less at random. The efficacy of these remedies in cases where real difficulty exists has not been clearly demonstrated. Where fuel is cheap and roasting can be done economically, a thorough oxidizing roast will, with but very few exceptions, solve the problem in the case of concentrates containing gold with little or no silver; but as I have elsewhere pointed out on a number of occasions, silver minerals are rendered refractory to cyanidation by an oxidizing roast. The chloridizing roast in such cases may be at times effective, but is not generally looked upon with favor by metallurgists on account of the possibility of loss of the precious metals. It is here sometimes suggested that the oil may be carbonized during roasting and thereby cause premature precipitation. This objection is not well founded, since with proper roasting it would be entirely consumed.

It seems to be the opinion among thoughtful metallurgists who have not been completely carried away by the wave of enthusiasm which has struck the world for flotation, that flotation will not to any considerable extent displace the cyanide process in the treatment of gold and silver ores, but, as previously pointed out, it will in some cases, perhaps a good many, prove more economical, and it will in many other cases prove a valuable accessory treatment to cyanidation. The reason that flotation will not displace the cyanide process is that most companies prefer to produce bullion. Furthermore, many gold and silver mines are situated far from smelting centers, which entails high freight rates and renders the shipment of concentrate out of the question. Some possibility, however, exists which tends to render such a prediction uncertain, and that is the possibility of obtaining an extremely high-grade concentrate such as could be directly melted with simple furnace equipment in much the same manner as cyanide precipitate. If we can conceive of this ever being accomplished, and it is not impossible, then cyanidation will certainly have a most formidable rival in flotation."

In talking on the same subject, Ernest A. Hersam, in *Metalurgical and Chemical Engineering*, June 15, 1916, points out that: "The greater number of oils allowed to remain in flotation concentrates are not chemically without action upon cyanide solution. Most oils have chemical qualities which also interfere

by the absorption of oxygen. The animal and vegetable oils are generally oxygen consuming. Combining with the oxygen at a critical time and place, when gold combines with the cyanide and when oxygen is needed, the extraction fails. Reduction and side reactions result in an attack upon the cyanide solvent. Particular interference is liable in the presence of the sulphide minerals soluble in alkali. Compounds as well, between mineral bases and oily acids, produce physical effects that retard the dissolving action."

The Late Estimate of Flotation.—A conclusion as to just what field is to be definitely filled by flotation is not easy to reach at this time. The period, reached in the early life of all metallurgical process, when it was considered to be a certain cure for all the previous metallurgical troubles of history, has been passed, and it is settling down to the comparatively prosaic routine of doing well the things that it can do. In the early part of 1918 it had been shown that flotation cannot do everything. Some copper ores can be leached with better results, from the business point of view, than be obtained by flotation. Some lead and zinc ores yield better profits when treated by ordinary gravity concentration. Some silver ores have demonstrated that cyaniding is still more profitable than flotation.

On the other hand, flotation has sometimes extended its field. The Cobalt ores of Ontario are being, in some cases, treated by flotation. The Buffalo mill is said to be getting good results from it. And flotation has been applied to some ores which were formerly cyanided. The shortage of cyanide caused by war conditions led to trials of flotation and in some instances the results were successful enough to induce the installation of permanent plants.

The great obstacle to the solving of flotation problems is still, as it has been from the beginning, litigation. The fear that any knowledge of their participation in flotation practice or experiment would lead to legal attacks has kept many operators silent, and consequently that free interchange of technical knowledge that leads to a more rapid solution of any technical problem, has been absent. Each operator has been obliged to go over ground that had already been covered by another—possibly many others—unknown to him. The publication of each step, as it was taken, would have made it possible to avoid this waste of effort.

It is hoped that the court hearings scheduled for the early part of 1918 will put a finish to this situation. Whatever the verdict may be, a decision that can be relied upon will be a relief to the operator in general, who will then know what conditions he will have to meet and whether or not flotation will assist him to earn profits. Excessive royalties will tend to restrict greatly the use of the process resulting in many cases in the anomalous situation of a step forward in the world's progress resulting in no profit to the community and failing to assist in the conservation of the public wealth. It is to be hoped that justice will emerge victorious in the flotation dispute.

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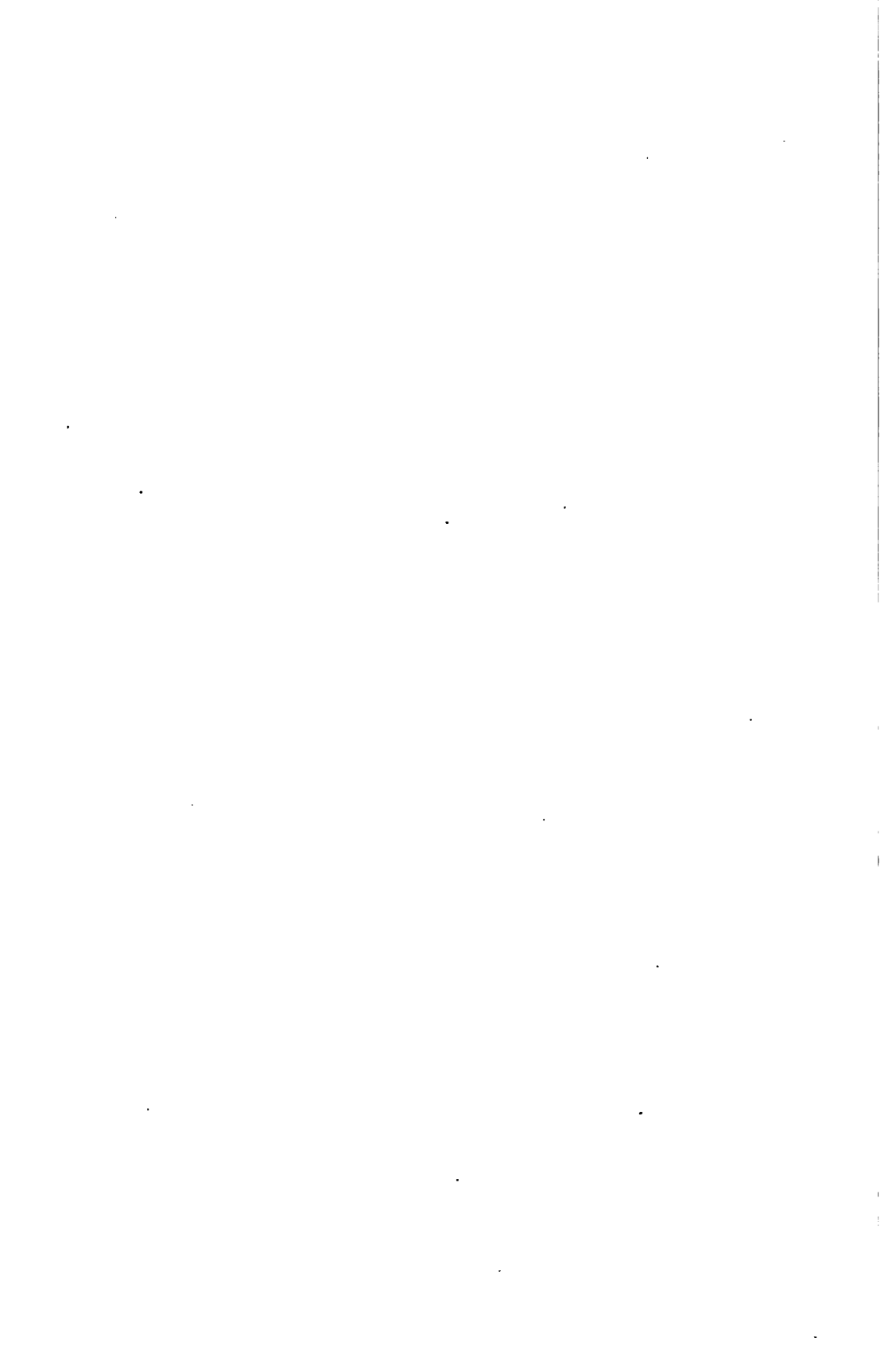
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